Investigating the Effect of Water Quality on the Adsorption of a Xanthate Collector in the Flotation of a Sulphide Ore

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ABSTRACT

Environmental concerns necessitate the recycling of process water within mining operations. On average, recycled water contains more dissolved solids than fresh water. Since water is used as both a transportation and process medium, it is expected that changes in its quality will affect plant processes.

Flotation is a process that is acutely sensitive to the immediate conditions of the system. Literature suggests that the efficiency of flotation separation is driven by the hydrophobicity that can be achieved by the mineral particles meant to be floated. The hydrophobicity is in turn driven by the adsorption of specialised reagents i.e. the collectors. Since collectors are added such that they adsorb at the liquid-particle interface, it stands to reason that changing the chemical composition of the aqueous phase will affect the collector adsorption, and hence the flotation response of target minerals.

In this study, a sulphide copper ore from the Zambian Copperbelt was floated in synthetic plant waters of varying ionic strengths, and with different concentrations of the collector sodium isobutyl xanthate (SIBX). The synthetic plant waters were prepared by adding varying concentrations of inorganic salts to distilled water in order to achieve process water compositions that reflect water compositions typically found in mining plants. Additionally, a nickel-copper ore from Lapland Finland was floated in the synthetic plant waters as well actual plant waters. To account for the latter ore's polymetallic nature, the collectors aerophine and sodium isopropyl xanthate (SIPX) were used sequentially.

The objective of the study was therefore to investigate the effect of water quality on collector adsorption in the flotation of sulphide ores.

The study showed that water quality has a quantifiable effect on SIBX and SIPX adsorption. The respective effects of water quality and collector adsorption on ore flotation could not be irrefutably decoupled. However, it could be concluded that of the tested waters, the copper thickener overflow was the least conducive to xanthate adsorption and valuable mineral recovery. On the other hand, collector adsorption was favoured by waters such as the raw and standard process. However, increased adsorption did not necessarily result in improved grades and recoveries.

The study further showed that in the case that the dissolved ionic species are identical, increasing the ionic strength of water yields a linear decrease in xanthate adsorption, and that recycling SIPX retained in flotation waters resulted in reduced separation selectivity.

GLOSSARY

°C	degrees Celsius
Au	gold
AsCu	acid-soluble copper
C1	first concentrate (with C2 being the second, C3 being the third, etc.)
CCC	critical coalescence concentration
cm	centimetres
CMR	Centre for Minerals Research
Cu	copper
DOW 200	DOW froth 200
Fe-Ti	iron-titanium
g	grams
g/mol	grams per mole
g/t	grams per ton
HG	high grade
ICP-OES	inductively coupled plasma optical emission spectrometry
I.S.	ionic strength
kg	kilograms
L	litres
L/mol.cm	litres per mole per centimetres
L/min	litres per minute
M	molarity
min	minutes
mg/L	milligrams per litres
mm	millimetres
Mm^3/a	mega cubic metres per year
ml	millilitres
mol/L	moles per litres

NF240	Nasfroth 240
Ni	nickel
nm	nanometres
Ore A	Kansanshi/Zambian high grade copper ore
Ore B	Lapland/Finnish nickel-copper ore
PAX	potassium amyl xanthate
Pd	palladium
PGE	platinum group element
PGM	platinum group metal
PIAX	potassium isoamyl xanthate
Pt	platinum
ppm	parts per million
rpm	revolutions per minute
SEX	sodium ethyl xanthate
SIBX	sodium isobutyl xanthate
SIPX	sodium isopropyl xanthate
TDS	total dissolved solids
UCT	University of Cape Town
UV/Vis spectroscopy	ultra violet-visible spectroscopy
μm	micrometres
μS/cm	micro Siemens
wt.	weight
XRF	X-ray fluorescence

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1. INTRODUCTION

1.1. Background

The scarcity of fresh water—exacerbated by a growing population's increasing demands, coupled with increasingly stringent environmental restrictions, has led to a need for industrial innovation in the usage of fresh water and the handling of waste waters. In particular, the mining industry has seen a surge of initiatives aimed at minimising fresh water usage by reusing and recycling process waters within mining operations (Liu et al., 2013). Froth flotation, a technology used to process much of the world's copper, zinc and lead-bearing minerals, among others, employs water as both a transport and processing medium (Bradshaw, 1997; Bunyak, 2000; Muzenda, 2010).

Based on empirical findings, Bleiwas (2012) estimates that a typical copper sulphide plant requires approximately 1.5 to 3.5 metric tons of water to process every ton of ore. Muzenda (2010) estimates that for a flotation circuit, the processed pulp comprises 80 to 85% water. The reuse and recycling of process waters leads to a change in chemistry. Often reclaimed from tailings facilities, thickener overflows and other dewatering systems, recycled waters contain, on average, more dissolved solids, and they are characterised by higher ionic strengths. Flotation is a physiochemical process, the efficiency of which is reliant on the differences in the surface properties of minerals. Previous studies have shown the tendency of recycled waters to alter mineral properties and the reactions occurring at mineral surfaces, which could in turn alter the overall process (Muzenda, 2010; Rao and Finch, 1989).

The manipulation of mineral surface properties is achieved by applying carefully developed reagent schemes, with collectors being used to enhance, typically, the hydrophobicity (and floatability) of valuable minerals (Bradshaw et al., 1998). Fuerstenau (1984) proposes that the efficiency of separation is dependent on the degree of hydrophobicity achieved by the target particles, with the hydrophobicity being dependent on the adsorption of the chosen collector. Collectors are added such that their adsorption onto minerals occurs at the mineral-water interface; in the absence of other electrolytes, collectors are the only adsorbing species. As such, species dissolved in solution influence collector adsorption (Fuerstenau, 1982b, Fuerstenau and Chander, 1986; Leja and Schulman, 1954). And so, the usage of recycled waters for such a water-intensive process has been noted to affect the separation of valuable from unwanted minerals, and the separation of valuable minerals from each other.

Despite the drawbacks, water recycling has the advantage of minimal fresh water use (especially in arid regions, and in regions where competition with sectors such as agriculture is extensive); it has the added advantage of reduced reagent consumption by recovering the unconsumed reagents retained in waste waters (Bleiwas, 2012; Levay et al., 2001; Slatter et al., 2009). Moreover, mining waste water facilities tend to be open and to interact with the surrounding environment. There is thus an interest to ensure minimal discharge of plant process waters, as they are typically contaminated with potentially toxic process reagents (Levay et al., 2001; Liu et al., 2013).

There is a growing interest to understand how recycled waters affect flotation. Numerous studies have elucidated the metallurgical performances of various ores and minerals floated in waters of varying properties, and with the reagent conditions varied within given limits. Though most have investigated the effect of single ions on metallurgical performance, few have investigated multi-ion systems, the better to mimic real-life recycled plant waters; fewer have simulated recycle scenarios, and even fewer than that have simulated both diverse *and* recycle systems.

The Centre for Minerals Research (CMR) has developed a standard synthetic plant water formula that reflected the typical water analysis of a PGM concentrator. Recent concentrator analyses show an increase in the ionic concentrations of recycled waters. As such, standard plant water concentrations are no longer typical of most PGM concentrators. The influence of ionic strength on flotation is thus studied by multiplying the ion level in the standard plant water (Corin et al., 2011; Wiese et al., 2005).

This study therefore aims to elucidate the effect of synthetic and actual plant water quality on the adsorption profiles of commonly used collectors onto sulphide ores, and to elucidate the effect of collector adsorption on flotation response. This was done by conducting batch flotation tests in simulated recycle scenarios on a Zambian high grade copper ore and a Finnish nickel-copper ore. UV/Vis spectrometry was used to determine the degree of collector adsorption.

1.2. Scope of the Study

Flotation is a complex process in which several factors interact; Klimpel (1984) summarises these as illustrated in Figure 1.1. This study focused on chemical parameters, more specifically, the adsorption of two xanthate collectors in actual and synthetic plant waters of differing ionic strengths. Different collector dosages were tested; in analysing the adsorption profiles, emphasis was placed on the amount of collector adsorbed and not the different xanthate species adsorbing, nor adsorption mechanism.

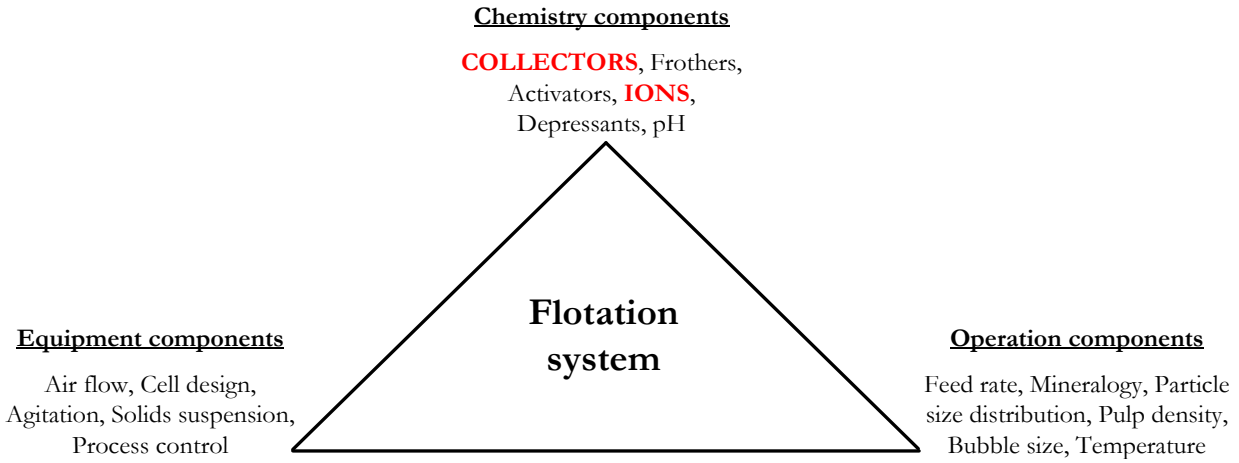

Figure 1.1: The different integral components that affect flotation systems, adapted from Klimpel (1984). The components relevant to this study are highlighted in red

Recycle scenarios were simulated by using water from a previous test to start another. The flotation performances of the ores were evaluated with the following as key indicators: the recovery of valuable minerals; the recovery of total solids, from which the recovery of gangue materials and the concentrate grades were determined; and the recovery of water, which was used along with total solids recovery as an indicator for froth stability and entrainment. Figure 1.2 summarises the scope of this study.

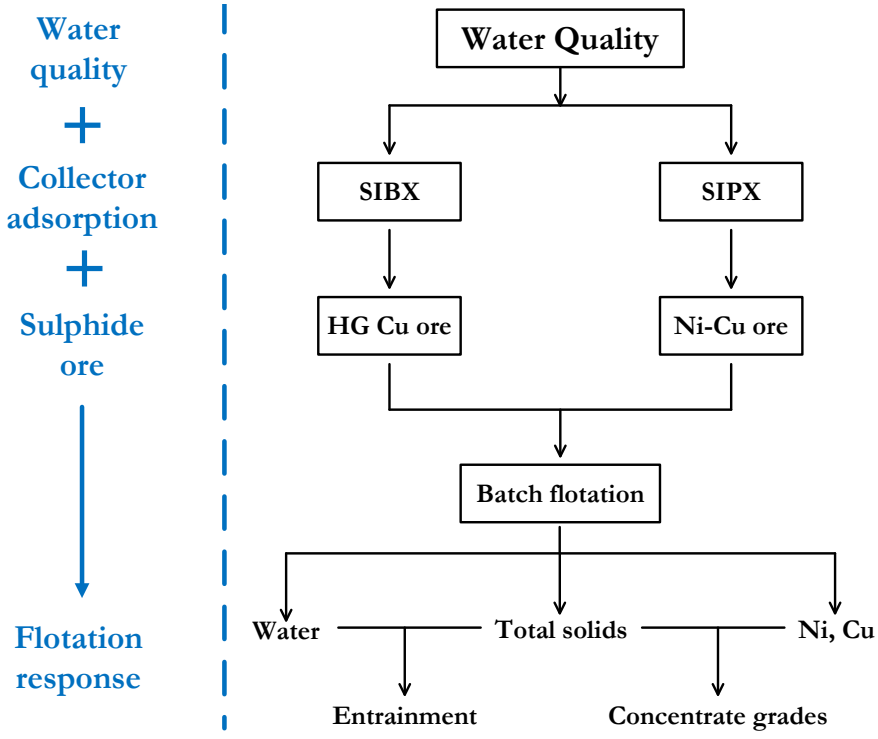

Figure 1.2: The scope of the study

2. LITERATURE REVIEW

2.1. Principles of Froth Flotation

Froth flotation is a physiochemical process of selectively separating minerals based on their surface properties. The valuable minerals are typically hydrophobic (water-repellent), while the unwanted particles, known as gangue minerals, are typically hydrophilic (water-loving). Flotation takes place in two distinct phases i.e. the pulp and the froth zone. When air is bubbled through the slurry, as illustrated in Figure 2.1, the hydrophobic particles attach to the air bubbles and float to the froth phase, where they are recovered and sent for further processing. The hydrophilic particles remain in the pulp and are collected in the tailings (Ata et al., 2004; Bradshaw, 1997; Taggart and Hassialis, 1946).

Thus, as reasoned by Finch (1995), flotation performs two elementary functions: the recovery of mineral particles by attaching them to bubbles, and the separation of these bubble-particle aggregates from the slurry; in which case, the former function is carried out in the pulp zone, and the latter is carried out in the froth zone.

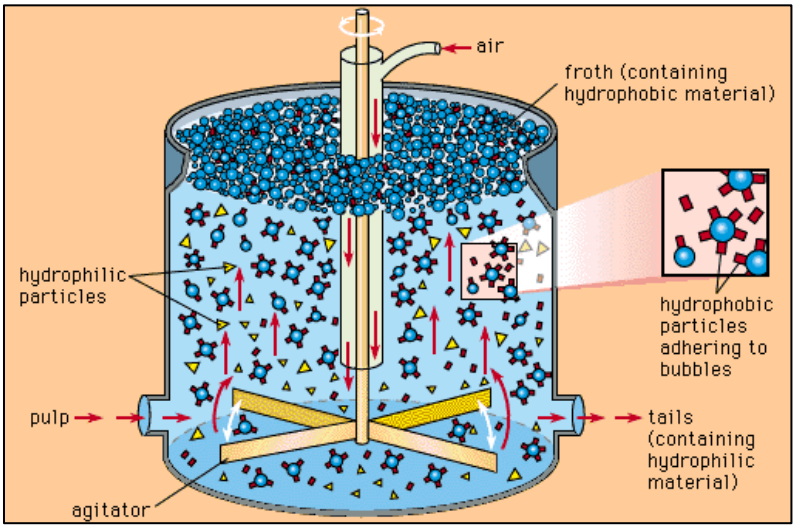

Figure 2.1: The fundamental components of froth flotation (Encyclopaedia Britannica, 2008)

The complete separation of gangue from valuable minerals in the pulp phase is often unachievable because minerals report to the concentrate either by true flotation or entrainment; and further, some gangue minerals are naturally floatable. As such, it is necessary to use reagents (depressants, activators, frothers and collectors) in the process in order to either enhance or reduce the hydrophobicity (and floatability) of the respective particles (Bradshaw et al., 1998).

First patented in 1877, froth flotation has since become among the most prominent processes for the recovery of sulphide minerals, processing much of the world's copper, zinc and lead-bearing minerals, among others (Bradshaw, 1997; Bunyak, 2000). In the majority of industrial cases, flotation takes place in mechanical cells; sometimes, columns are used. In a flotation cell, appropriately sized and liberated solid particles of differing commercial value are suspended in water, treated with flotation regents, and subjected to aeration (Finch, 1995; Ghosh, 2012; Harris, 1982; Leja and Schulman, 1954).

2.1.1. The Pulp Phase

Froth flotation exploits the differences in the surface properties of commercially valuable and valueless minerals that are contained within the same ores. In order to liberate the valuable from the valueless minerals, the ore is crushed and milled (typically, in wet conditions) to a particle size distribution where the coarse fraction is characterised as being greater than 75 μm and the fine fraction as being below 10 μm. The slurry is then fed to a flotation circuit (Loveday and Brouckaert, 1995; Senior et al., 1995). Bleiwas (2012) reports that on an industrial scale, for a conventional copper sulphide processing plant, nearly 1.5 to 3.5 metric tons of water is required to process every ton of ore from the crushing to the flotation stages. Muzenda (2010) estimates that a flotation cell is typically comprised of 80 to 85% water by weight fraction, and 15 to 20% solids.

In the pulp phase, the mineral particles meant to be floated attain hydrophobicity by interacting with the added collector; subsequently, the hydrophobic particles attach to bubbles and float to the froth phase; this is termed true flotation. The gangue minerals, being naturally hydrophilic or having attained hydrophilicity through interaction with a depressant, are hydrated and hence separated from the valuables. The efficiency of separation is dependent on the degree of hydrophobicity achieved by the valuable particles. The hydrophobicity is in turn dependent on the adsorption of the chosen collector onto the surfaces of the particles meant to be floated (Fuerstenau, 1984; Taggart and Hassialis, 1946).

Furthermore, the efficiency is dependent on the probability of contact between air bubbles and the hydrophobic particles, and this is in turn reliant on the relative sizes of the particles. For optimal contact, the respective bubble and particle diameters are comparable to each other. When the bubbles are large (relative to the particles), the probability of contact is reduced since water flowing around the bubbles can sweep off the particles before contact is achieved. When the bubbles are small, their buoyancy might not be sufficient to lift the particles to the froth (Ghosh, 2012; Kawatra, 2009; Whelan and Brown, 1956).

The adsorption of a collector onto the mineral surface often occurs at the mineral-water interface. Unlike frothers, collectors are generally electrolytes; in the absence of other electrolytes, collectors are the only species adsorbing onto the mineral surfaces. Factors such as pulp potential, chemical composition and pH thus affect the adsorption efficiency (Fuerstenau, 1982b).

Oxidising pulp environments improve sulphide hydrophobicity by rendering mineral surfaces sulphur-rich and non-polar, and by improving collector adsorption (Hintikka and Leppinen, 1995; Kirjavainen et al., 2002). In extremely reduced potentials, xanthates cannot optimally adsorb. For chalcopyrite, it has been shown that the mineral surface is rendered hydrophilic in reducing environments, and that the reducing atmosphere caused by steel grinding is sufficient to suppress particle floatability. Floatability can be re-established by raising the potential by aeration or the addition of oxidants. However, the restoration is often limited for coarser size fractions (Fuerstenau et al., 1968).

Gangue minerals suspended in the pulp phase do not interact with the air-water interface, hence their recovery is typically attributed to entrainment. Unlike true flotation, entrainment, which is the mechanical carrying of particles through the movement of water from the pulp and into the froth phase, is unselective. Particles below 38 μm in size are more likely to be recovered by entrainment, and their rate of recovery is equal to the rate of water recovery. Adequately coarse particles tend to settle rapidly enough to avoid being entrained to the froth. Finer particles, on the other hand, settle more slowly and thus, they are easily carried into the froth by entrainment, and once there, are not easily drained back into the pulp (Ata et al., 2004; Wills and Napier-Munn, 2006).

In addition to entrainment, gangue minerals can be carried to the froth zone by being locked to valuable hydrophobic particles, or by behaving as floatable-gangue composite particles (Becker et al., 2009; Corin et al., 2011). For example, "*In the case of coal [flotation], much of the [gangue] pyrite consists either of sub-micron pyrite grains that are never liberated from the coal, or of pyrite particles whose surfaces consist primarily of coal, and therefore behave as if they were coal particles.*" Similarly, valuable minerals can remain in the pulp as a result of being locked in hydrophilic particles, or behaving as composite particles (Kawatra, 2009).

2.1.2. The Froth Phase

The froth phase (or zone) is the air-water interface in which the floated minerals are recovered to the concentrate (Harris, 1982). Froth stability is essential for optimum mineral recovery. If the froth is not adequately stable, the floated particles can be drained back into the pulp as a result of the bubbles breaking prematurely i.e. before the froth has flown over the cell lip or been scraped off. However, if

the froth is too stable, the selectivity of the process can be hindered. Additionally, the bubble surface area in the froth phase can affect mineral recovery in the following way: increased bubble surface area yields higher particle flotation rates, but at the same time, more water is carried into the froth as the film between the bubbles. This will in turn increase the recovery of entrained particles recovered to the froth; thus, the recovery of gangue minerals will be increased (Kawatra, 2009; Slatter et al., 2009).

As the froth ages, slurry drains from the froth and a portion of the entrained gangue drops back into the pulp. The remainder of the entrained gangue is carried to the concentrate. Moreover, valuable mineral particles with weaker contact angles are recovered at lower rates than those with higher contact angles (discussed in greater detail in Section 2.3.4.1) (Harris, 1982; O'Connor and Mills, 1995).

In addition to froth stability, factors that influence drainage include gangue mineral density, particle size, composition and concentration. Hydrophobic particles accelerate the rate of liquid drainage, thus increased concentrations of hydrophobic particles in the froth reduce the recovery of hydrophilic particles and water in the concentrate. This suggests that froth stability is strongly dependent on the type and concentration of particles present (Bruckard et al., 2011; Corin et al., 2011; Lovell 1982).

2.2. Flotation Reagents

Flotation reagents are used in the flotation process to optimise the degree of separation efficiency i.e. the recovery of valuable minerals, and the depression of non-valuables. Some of the earliest flotation reagents were selected oils whose properties were, as suggested by Dean and Ambrose (1944), fortuitous. Further postulated by Dean and Ambrose is the following: "*In order to function, an oil must have an active group—one that differs from the remainder of the molecule in its affinity for water or the mineral particle.*"

Over the years, specialised chemicals have been developed to perform specific functions in enhancing differences in the surface properties of minerals particles (Bulatovic, 2007). However, flotation is an exceedingly complex process, wherein several interactions occur between the reagents. Moreover, the minerals present in each ore react differently to these interactions, making it difficult to quantify the precise behaviour of each of the reagents (Bradshaw et al., 1998; Wiese et al., 2006). Reagents can be divided into three types: frothers, collectors and regulators, with the latter type comprising activators and depressants (Lovell, 1982).

2.2.1. Frothers

Frothers are non-ionic, heteropolar, surface active molecules that are added to provide a stable liquid-air interface such that the floated particles do not fall back into the pulp before being recovered in the concentrate. Frothers increase the dispersion of air through the flotation cell, reduce the coalescence of individual bubbles in the pulp phase, and decrease the rate at which the bubbles rise to the froth phase, which in turn increases the possibility of bubble-particle contact (Harris, 1982; Lovell, 1982).

Certain frothers are effective at certain pH ranges; as such, frothers can be categorised according to their effectiveness in different pulp pH values, summarised in Table 2.1 (Bulatovic 2007).

Table 2.1: Basic frother groups as classified by their effectiveness in different pulp pHs (Koshdast and Sam, 2011)

Acidic	Neutral	Basic
Phenols	Aliphatic alcohols	Pyridine base
Alkyl sulfonates	Cyclic alcohols and natural oils	
	Alkoxy paraffins	
	Propylene glycol ethers	
	Polyglycol ethers	
	Polyglycol glycerol ethers	

The frothing properties of acidic frothers are reduced with increasing pulp pH. Over the years, environmental restrictions have limited the application of acidic frothers. The most important group of frothers is the neutral. Being effective in both acidic and alkaline ranges, these frothers are used in the flotation of base-metal ores, oxidic minerals and industrial minerals. Of the neutral frothers, the most widely used are the polyglycol ethers, with several variations being produced by different manufacturers (such as the Dowfroths produced by the Dow Chemical Company and the Aerofroths produced by Cyanamid) (Bulatovic, 2007; Koshdast and Sam, 2011).

The performance of polyglycol ethers, with regard to frothing ability and selectivity, is dependent on the molecular weight and hydrocarbon chain length of the chosen frother. Frothers with higher molecular weights produce more persistent froths, however, this group also tends to be less selective than their lower molecular weight counterparts (Koshdast and Sam, 2011). Another parameter used to characterise frothers is their critical coalescence concentration (CCC) values. As the concentration of frother in solution increases, bubble coalescence decreases until, at a given concentration—the CCC, illustrated in Figure 2.2, coalescence stops. CCC values for stronger frothers occur at lower concentrations than for weaker frothers (Cho and Laskowski, 2002).

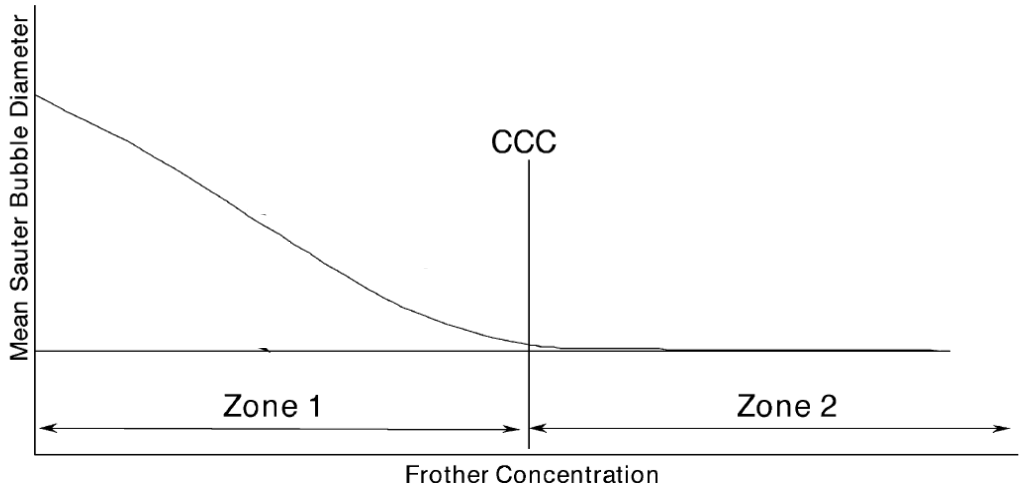

Figure 2.2: The effect of frother concentration on bubble size, adapted from Cho and Laskowski (2002)

Due to the complexity of the flotation process, the mechanisms of the frother action are not entirely independent of other reagents; nor are they independent of other factors such as type and size of the floated and gangue minerals and the presence of other chemical species present in the froth phase (Lovell, 1982). But synthetic frothers such as the alcohol and polypropylene glycol subgroups have insignificant collector properties, making it possible to regulate the frother dosage without also changing the collector dosage (Kawatra, 2009).

2.2.2. Modifiers

Modifiers are added in the pulp phase to either enhance or hinder the adsorption of collectors to mineral particles. For example, in cases where collectors do not adequately attach to the valuable mineral particles, activators are added to enhance the collector adsorption. Conversely, depressants are used to achieve one of two effects; in the first case, they are used to reduce the floatability of gangue such as talc, which is a naturally hydrophobic mineral that requires no collector to float. Secondly, in cases where the collector, in addition to adsorbing to valuable minerals, adsorbs onto gangue minerals, depressants are added to enhance the hydrophilicity of the gangue and thus ensure that only the valuable minerals float (Becker et al, 2009; Bradshaw et al., 1998; Lovell, 1982).

Depressants can be either organic or inorganic, with the latter group largely growing obsolete as a result of the compounds' toxicity and ensuing environmental restrictions. The organic group of depressants (in the form of the two polysaccharides modified guar gum (guar) and carboxymethyl cellulose (CMC)) is the more commonly used for sulphide mineral flotation (Bulatovic, 2007). Activators are inorganic compounds with examples being as follows (Lovell, 1982):

- Copper sulphate, which produces a Cu^{2+} ion for the activation of sulphide minerals.
- Sodium sulphide (Na_2S) and hydrosulphide (NaSH) for the sulphidisation of tarnished oxide minerals and carbonates.

2.2.3. Collectors

Collectors are heterogeneous molecules that contain an inorganic, active group, and a non-ionic, hydrocarbon chain (a general molecular structure is illustrated in Figure 2.3). They are added to the pulp phase to, ideally, form a hydrophobic layer on a specific mineral or group of minerals (Dean and Ambrose, 1944). In the adsorption process, the inorganic group, which is polar and hydrophilic in nature, attaches to the mineral surface, while the hydrocarbon chain, which is orientated towards the water phase, renders the mineral surface hydrophobic and subsequently attaches to an air bubble (Garrels and Christ, 1990; Lovell, 1982).

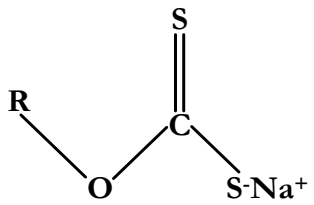

Figure 2.3: The general molecular structure of a heterogenous collector, where 'R' represents the hydrocarbon radical, adapted from Lovell (1982)

The classification of collectors into their respective groups is dependent on the degree to which they dissociate when dissolved in water. The groups are as follows: ionic, cationic and non-ionic. The ionic group consists of heteropolar organic molecules which, when in water, can assume the property of either an anion or a cation. Further, depending on whether they are more suited to oxide or sulphide mineral flotation, anionic collectors can be classed into oxhydryl or sulphydryl subgroups. Non-ionic collectors consist of two groups, the first of which contain bivalent sulphur, and the second, which contain non-polar hydrocarbon oils. Finally, the cationic group consists largely of amines in which the hydrocarbon radical is protonised (Bulatovic, 2007; Dean and Ambrose, 1944).

The most widely used collectors in flotation belong to the anionic group. Of these, collectors belonging to the oxhydryl subgroup are primarily used for the flotation of silicates, oxide minerals and carbonate materials. The sulphydryl subgroup is the most commonly used family of collectors in the flotation of sulphide minerals; more specifically, xanthates and dithiophosphates are the most commonly used (Rao, 2003; Lovell, 1982).

2.2.3.1. Oxhydryl Collectors

Oxhydryls are less selective than sulphydryl collectors; however, their selectivity has been shown to improve when they are used in conjunction with certain modifiers and when the pulp pH is carefully controlled. Figure 2.4 depicts the general molecular substructures of two oxhydryl families: a carboxylate (under which falls collectors such as sodium oleate and oleic acid, among others), and an alkyl sulphate (under which fall sodium alkyl sulphates). Other oxhydryls include: sulphonates, hydroxamates and phosphoric acid esters (Bulatovic, 2007).

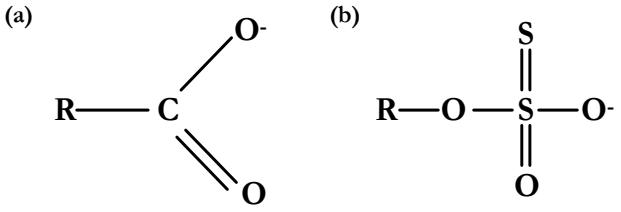

Figure 2.4: The general molecular substructures of two oxhydryl collector families; (a) a carboxylate, (b) an alkyl sulphate. In each case, 'R' represents a hydrocarbon radical group, adapted from Bulatovic (2007)

2.2.3.2. Sulphydryl Collectors

Sulphydryl collectors are more commonly known as thiols, and can be further divided into the following main chemical families: dithiocarbonates (better known as xanthates), dithiocarbamates and dithiophosphates. A relatively new family—the dithiophosphinates, has been added to the thiol group (Cytec Solvay Group, 2018; Rao, 2003).

Given their high selectivity for sulphides and general non-affinity for non-sulphide gangue minerals, xanthates have emerged as the most important thiol family for sulphide flotation. More specifically, short chain xanthates are preferred for sulphide flotation because the large sulphide ion in the mineral (compared to the oxygen ion in oxide minerals) does not typically form hydrogen bonds. As such, sulphide minerals tend to be naturally less hydrophilic than oxygen-bearing minerals (Fuerstenau, 1982d; Lovell, 1982; Rao, 2003). Figure 2.5 illustrates some of the most usually used xanthates (of varying alkyl length and configuration).

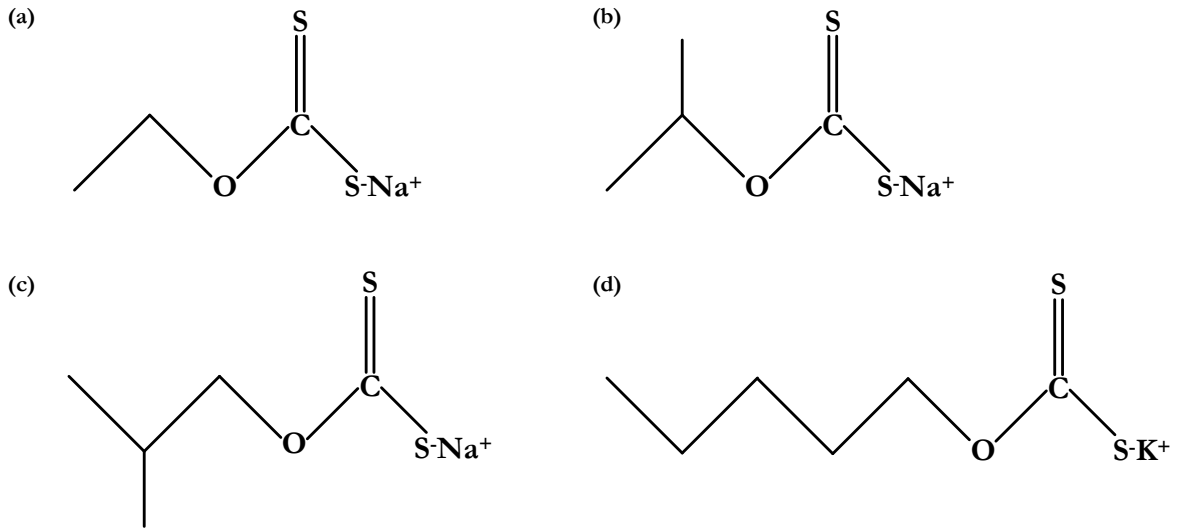

Figure 2.5: The molecular structures of industrially used thiols of varying hydrocarbon chain length: (a) sodium ethyl xanthate (SEX), (b) sodium isopropyl xanthate (SIPX), (c) sodium isobutyl xanthate (SIBX), and (d) potassium amyl xanthate (PAX), adapted from Bulatovic (2007); Dean and Ambrose (1944); Lovell (1982)

Unlike dithiophosphates, the hydrocarbon radicals in dithiophosphinates are directly linked to the phosphorus atom (Bulatovic, 2007). An example of one of the most widely used dithiophosphinates—aerophine, is illustrated in Figure 2.6.

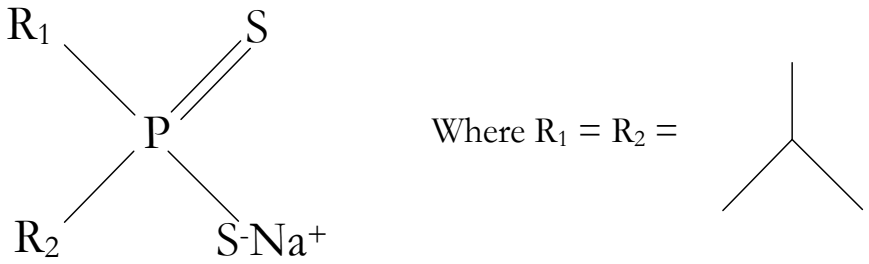

Figure 2.6: The molecular structure of sodium dithiophosphinate (most commonly known as Aerophine 3418A), adapted from Bulatovic (2007)

The general adsorption behaviours of the different thiol families are somewhat different, and as such, the different collectors are used for different separations. For example, aerophine, though not as strong as the xanthates, is preferred for its selectivity, and thus applied in the flotation of complex sulphide ores (in the recovery of such fast-floating minerals as chalcopyrite) (Cytec Solvay Group, 2018; Güler et al., 2004). As a general rule, collectors are surface active compounds; however, most, especially short-chain thiol-type collectors, are not active at the liquid-air interface (Bradshaw, 1997, Lovell, 1982). Their typical adsorption behaviours, and the factors affecting these behaviours, are discussed in greater detail henceforth.

2.3. Adsorption Mechanisms of Collectors

The adsorption of collectors occurs by physical or chemical means, termed physisorption and chemisorption, respectively. As suggested by empirical adsorption isotherms and electrokinetic measurements (illustrated in Figure 2.7), at low concentrations of collector in solution, collector ions adsorb as individual ions in the electrical double layer (further discussed in Section 2.3.4), while at higher concentrations they associate at the liquid-mineral interface into aggregates known as hemimicelles. But even at high collector concentrations, electrostatic effects are effective (Fuerstenau and Chander, 1986; Taggart and Hassialis, 1946).

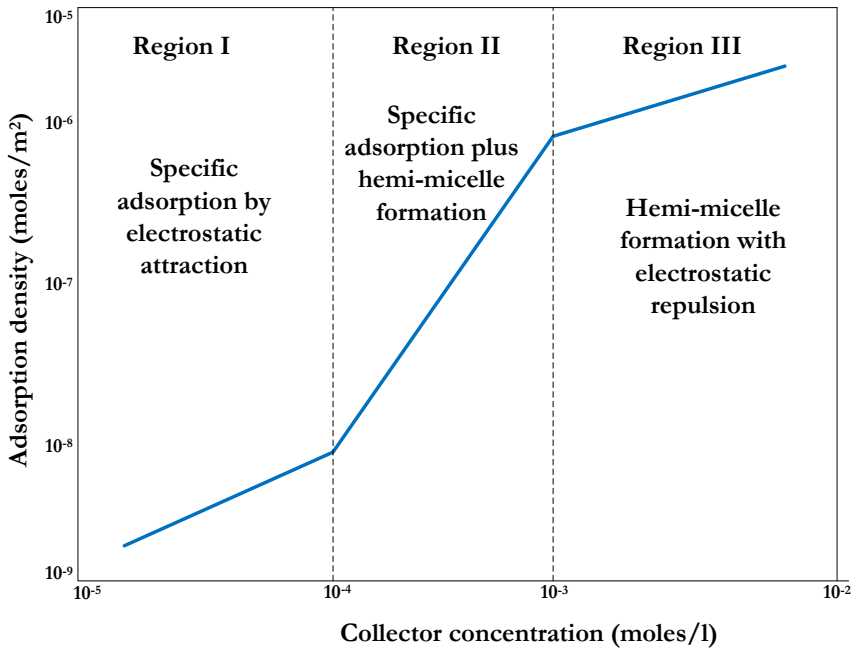

Figure 2.7: The adsorption density of a collector relative to its initial concentration in solution, adapted from Fuerstenau (1982e)

As stated by Fuerstenau et al. (1970): "*A hydroxyl complex adsorbs on the mineral surface first, resulting in a positively charged surface. Negatively charged collector ions could then be attracted to the surface. However, other findings suggest that [in some cases] an aqueous species of the metal hydroxyl complex and collector is the species adsorbing at the liquid-mineral interface.*"

2.3.1. Physisorption

Physisorption occurs when collectors are adsorbed only through electrostatic attraction and hydrophobic bonding, for example, Van der Waals interactions between hydrocarbon chains. (Fuerstenau and Chander, 1986). Figure 2.8 illustrates how anionic collectors may adsorb in the electrical double layer. The different regions in Figure 2.8 correspond to those in Figure 2.7.

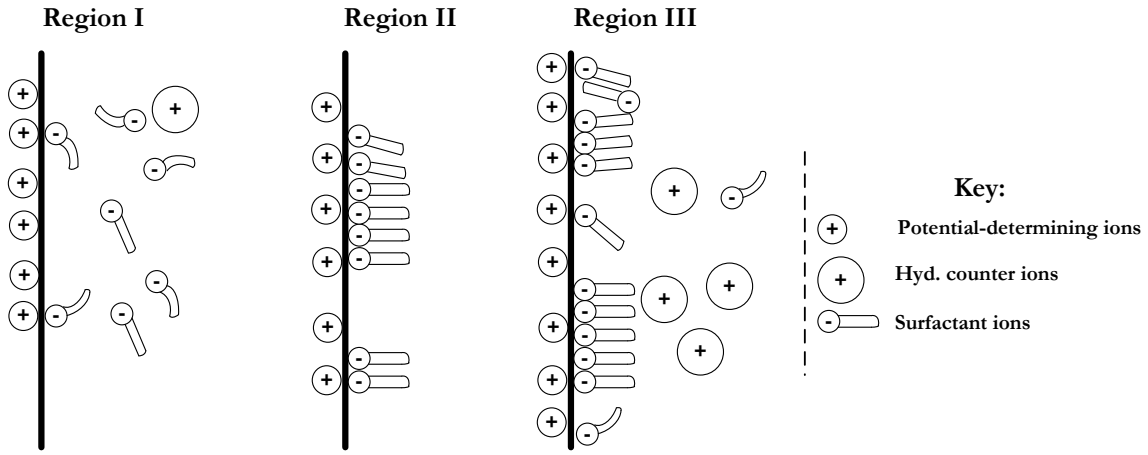

Figure 2.8: Illustration of adsorbed surfactant ions in the double layer, depicting adsorption as individual ions and as associated hemimicelles, adapted from Fuerstenau (1982e)

2.3.2. Chemisorption

Chemisorption occurs when collectors form covalent bonds with metal ions on the mineral surface. The selective flotation of some minerals (or groups of minerals) can be achieved if there exists a specific collector-mineral reaction which renders that specific mineral hydrophobic, forming strong adsorption films that are difficult to remove (Fuerstenau and Chander, 1986; Raju and Forsling, 1991).

The electrochemical theory of adsorption proposes that xanthates adsorb onto sulphide surfaces via the formation of a metal-xanthate compound. The first step of the mechanism is the oxidation of the mineral surface (Equation 2.1); this is then followed by the chemical reaction between the collector and the mineral surface (Equation 2.2). The overall adsorption mechanism can be expressed by Equation 2.3, where ***MS*** is the metal sulphide, ***X*** is the xanthate collector and ***MX_2*** is the metal-xanthate complex (Fuerstenau et al., 1968; Yoon and Basilio, 1993):

$$MS \rightarrow M^{2+} + S^{\circ} + 2e^{-} \qquad 2.1$$

$$M^{2+} + 2X^{-} \rightarrow MX_2 \qquad 2.2$$

$$\therefore\ MS + 2X^{-} \rightarrow MX_2 + S^{\circ} + 2e \qquad 2.3$$

The concentration of the metal-xanthate complex has been noted to affect the flotation of some sulphide minerals. For example, the presence of multiple layers of lead-xanthate increase the floatability of galena. In addition to chemisorption and metal-xanthate formation, xanthates adsorb onto sulphide minerals via the catalytic oxidation of the collector into a dimer known as dixanthogen. The formation of dixanthogen is strongly dependent on pH values of solution as well as pulp

potentials. Under varying pulp phase conditions, chalcopyrite has been observed to float either via the formation of CuX or dixanthogen, or a combination of the two (Yoon and Bailio, 1993).

On the other hand, pyrite has been observed to float only through dixanthogen formation, while dixanthogen does not form on chalcocite surfaces. Pyrite and chalcopyrite are noble and don't liberate sufficient metal ions to form metal-xanthate at lower potentials. At high potentials, chalcopyrite oxidises to liberate sufficient copper ions to form CuX (Yoon and Basilio, 1993). The oxidation of xanthates into dimers is expressed by equations 2.4 and 2.5, where $\boldsymbol{X_2}$ is the dixanthogen (Fuerstenau et al., 1968; Fuerstenau, 1982c):

$$2X^- + {}^1/_2\, O_2 + H_2O \rightarrow X_2 + 2OH^- \qquad 2.4$$

$$2X^- + M^{2+} \rightarrow MX(s) + {}^1/_2\, X_2 \qquad 2.5$$

2.3.3. Thermodynamic Models for Collector Adsorption

Xanthate collectors typically coat mineral particles through multilayer chemical adsorption. In general, the rate of adsorption is fast, with the reversible equilibrium reached in seconds. However, empirical evidence has demonstrated that the adsorption of xanthates onto galena is continuous, neither reaching completion nor equilibrium. This suggests that beyond such variables as pulp temperature, collector concentration, pH, etc., the extent of reaction can be described with reference to reaction kinetics, where, in the initial stages of contact between the mineral and the collector, the rate equation is exponential (Equation 2.6). In Equation 2.6, $\boldsymbol{a}$ and $\boldsymbol{\beta}$ are constants, while $\boldsymbol{q}$ is the amount of adsorbed collector at time $\boldsymbol{t}$ (Finkelstein and Lovell, 1972).

$$\frac{dq}{dt} = \beta . \exp(-aq) \qquad 2.6$$

However, for sufficiently low concentrations of solute i.e. the surfactant, a relationship can be derived between the amount (of solute) adsorbed at the mineral interface and the activity of the adsorbate. This is termed an adsorption isotherm, and, depending on the assumptions made for each mineral-collector system, can be expressed by one of several models. For example, because of the electron exchange between the surfactant and the mineral surface, true chemisorption is restricted to monolayer coating. Thus, in this case, when multilayer adsorption is less probable, the adsorption process can be modelled by the Langmuir monolayer equation (Equation 2.7) (Raju and Forsling, 1991; Rao, 2003).

$$\frac{C_{eq}}{x/m} = \frac{1}{k \cdot (x/m)_{max}} + \frac{C_{eq}}{(x/m)_{max}} \qquad 2.7$$

In Equation 2.7, C_{eq} is the concentration of the collector at equilibrium; x is the amount of adsorbed collector, k is the Langmuir isotherm constant, and $(x/m)_{max}$ is the maximum amount of adsorbed collector per mass of mineral (Rao, 2003).

2.3.4. Factors Affecting Collector Adsorption

Given the distinctly physiochemical nature of flotation, it can be expected that variations in conditioning are likely to yield variations in the process performance. Firstly, the properties of the mineral-water interface affect flotation by controlling two factors. The first is the interaction of water molecules with the mineral surface, and the second is the electrical double layer at the interface (illustrated in Figure 2.9). An electrical double layer forms when there is a separation of electrical charge at an interface; that is, when there exist separate layers of positive and negative charge, but with the entire system being electrically neutral (Fuerstenau, 1982a; Lovell and Finkelstein, 1972).

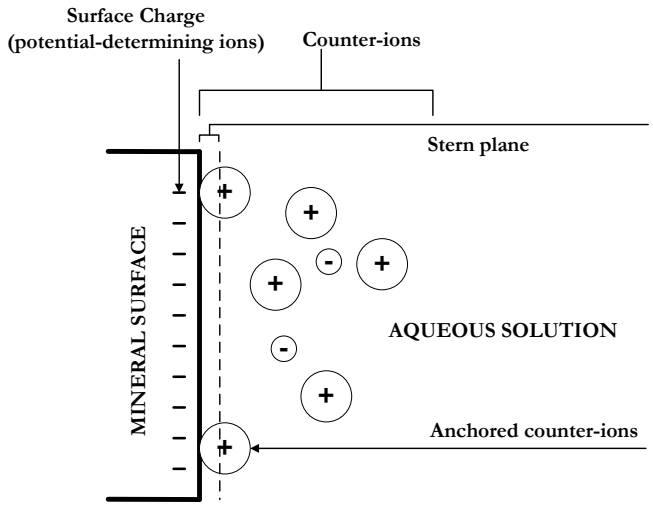

Figure 2.9: The electrical double layer of a negatively-charged mineral surface, adapted from Fuerstenau (1982a)

Ions that transfer between different phases and establish the electrical double layer are called potential-determining ions. These ions are products of hydrolysis; when minerals come into contact with water, a polar substance, ions are transferred from the mineral and into the solution. As such, each mineral has a unique set of potential-determining ions. Hydrolysis products of dissolved metal salts can also act as potential-determining ions. The surface charge of the mineral is thus determined by the adsorption density of the potential-determining ions (Fuerstenau, 1982a).

The electrical double layer affects flotation as follows: the sign of the surface charge affects the adsorption of other ions onto the surface. Ions charged oppositely to the surface function as counter-ions, thus maintaining the overall electro-neutrality. Some ions demonstrate surface activity and electrostatic attraction, and thus adsorb in the double layer due to covalent and hydrophobic bond formation. Collectors typically act as surface-active counter-ions. Overall, the electrical double layer affects the flotation process as follows (Fuerstenau, 1982a):

- The sign and size of the surface charge drives adsorption of physically-adsorbing reagents.
- A high surface charge can inhibit the adsorption of chemically-adsorbing collectors.
- It controls the flocculation and dispersion of mineral suspension.

Collectors are thus classed into two types: surfactants which chemisorb or chemically react at the surface, and long-chain ionic surfactants which physically adsorb as counter-ions. The former class comprises xanthates, which chemically react with the surface to form metal-collector complexes on the mineral surfaces as well as in the bulk solution, as is often the case in sulphide mineral flotation. The latter class is generally used in oxide mineral flotation, and is characterised by strong hydrocarbon-hydrocarbon chain interaction in the double layer (Fuerstenau, 1982b).

2.3.4.1. Bubble-Particle Attachment

The degree of attachment between a mineral particle and a bubble greatly affects flotation efficiency. This parameter is expressed in terms of a contact angle, depicted in Figure 2.10 and defined by Young's equation (Equation 2.8) (Fuerstenau, 1982b; Taggart and Hassialis, 1946).

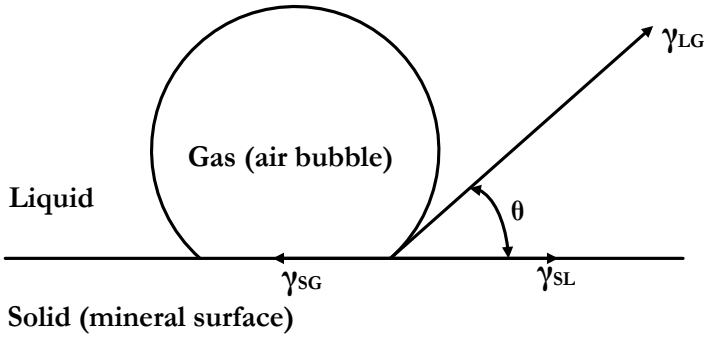

Figure 2.10: The equilibrium contact between a bubble and a solid particle, adapted from Fuerstenau (1982b); Kawatra and Eiesele (2001)

In Figure 2.10, θ is the contact angle, measured through the liquid; γ_{SG}, γ_{SL} and γ_{LG} are the tensions of the solid-gas, solid-liquid and liquid-gas interfaces, respectively. The change in the free energy

associated with the replacement of a unit area of the solid liquid interface by the solid-gas interface is given by Dupre's equation (Equation 2.9). Thus, the free energy change can be obtained by combining Young and Dupre's equations to yield Equation 2.10 (Fuerstenau, 1982b).

$$\gamma_{SG} = \gamma_{SL} + \gamma_{LG} \cdot \cos\theta \quad 2.8$$

$$\Delta G = \gamma_{SG} - (\gamma_{SL} + \gamma_{LG}) \quad 2.9$$

$$\therefore \Delta G_S = \gamma_{LG} \cdot (\cos\theta - 1) \quad 2.10$$

Thermodynamically, a process will spontaneously occur if the free energy decreases; hence, for bubble adhesion to take place, $\Delta G_S < 0$ in the liquid medium. Thus, for any solid-liquid system where $\theta > 0$, bubble adhesion will occur (Fuerstenau, 1982b). Furthermore, Taggart and Hassialis (1946) assert: "*It has been established beyond the possibility of successful contradiction, that a mineral in a given chemical environment in an air-bubble-contact-angle test that gives a contact angle of 45° or over, with an induction time of 30 seconds to a minute, will float in the same chemical environment in a flotation cell.*"

As previously discussed, optimal contact between bubbles and particles is achieved when their respective diameters are comparable; with the probability of contact decreasing when the bubbles are too large or too small (Kawatra, 2009; Whelan and Brown, 1956).

2.3.4.2. The Effect of Pulp pH

The adsorption of xanthates onto sulphide minerals is strongly influenced by pulp pH. For example, in a pyrite flotation system, the dixanthogen species responsible for pyrite flotation is stable until a pH value of 11, above which the dimer becomes unstable and pyrite becomes depressed. On the other hand, chalcocite is floated at a higher pH because the cuprous-xanthate species responsible for flotation is stable in that range. In general, xanthate collectors decompose with a decrease in pH, and are stable at pH values ranging from 8 to 13 (Fuerstenau, 1982c; Fuerstenau et al., 2007).

Furthermore, minerals tend to be positively charged when in acidic solutions, and negatively charged when in alkaline solutions. And so, for each mineral there exists some theoretical pH value at which the surface charge changes from positive to negative, and thus some optimal pH value suitable for collector adsorption. In this way specific minerals can be progressively separated from each other (Fuerstenau et al., 1985; Kawatra, 2009). Figure 2.11 is an illustration of the behaviour of galena, chalcopyrite and pyrite, floated with a dithiophosphate at different pH values.

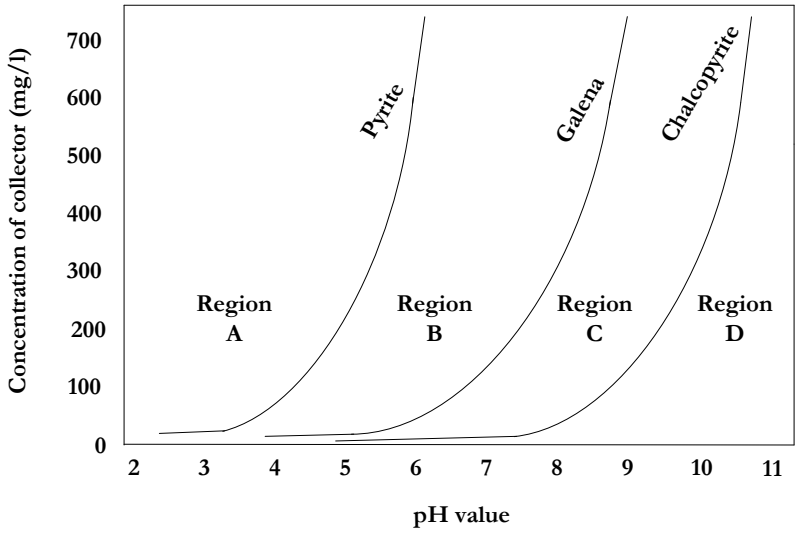

Figure 2.11: The sulphydryl collector-induced flotation responses of sulphide minerals in different pH ranges, with sodium diethyl dithiophosphate as the collector, adapted from Fuerstenau (1982c); Wark and Cox (1934)

In region D, no collector adsorption occurs, and the minerals are not rendered sufficiently hydrophobic to float. As the pH is decreased into region C, chalcopyrite becomes sufficiently hydrophobic to float. As the pH is further decreased into region B, adsorption is further increased and galena, in addition to chalcopyrite, will float. And finally, all three minerals will float in region A (Fuerstenau, 1982c; Fuerstenau et al., 1985).

2.3.4.3. The Effect of Hydrocarbon Chain Length and Configuration

Hydrocarbon chain lengths and configurations are theorised to influence the required collector concentration to render sulphide minerals hydrophobic. The adsorption of long chain collectors requires lower concentrations in order to saturate all surface-active sites on the mineral. However, high adsorption density does not necessarily result in increased hydrophobicity and floatability. This is because long-chain adsorption may result in secondary adsorption by hydrophobic bonding, which leads to the formation of micelles and the exposure of polar sites (Makanza et al., 2008).

In a study conducted by Plaksin et al. (1954), the authors showed that in the flotation of arsenopyrite, ethyl xanthate was a weaker collector than both amyl xanthate and sodium diethyl dithiophosphate. It was theorised that the presence of the two ethyl radicals in the dithiophosphate, or the composition of the molecule, enhanced the effectiveness of the collector. Amyl xanthate was shown to react more slowly with the mineral than ethyl xanthate. But the longer hydrocarbon chain of the amyl xanthate resulted in a higher total recovery of the mineral. And when compared to the ethyl xanthate, the amyl xanthate yielded maximum flotation rates that were achieved in a longer flotation time. But overall,

the fastest maximum rate was achieved in flotation with the phosphate, suggesting a superior initial reaction of the phosphate with arsenopyrite.

2.3.4.4. The Effect of Collector Concentration

For a given range of collector concentration, as the amount of collector dosed to the system is increased, the amount of adsorption is also increased and thus the mineral recovery improved. This relationship reaches a plateau phase, along which increasing the collector concentration neither increases the amount of adsorbed collector, nor significantly improves mineral recovery. Beyond this point, increasing the collector concentration results in decreased mineral recovery (Wang, 2016). This is exemplified by the concentration and adsorption variance of 8-hydroxyquinoline, and the resultant effects on the flotation of wolframite, as illustrated in Figure 2.12.

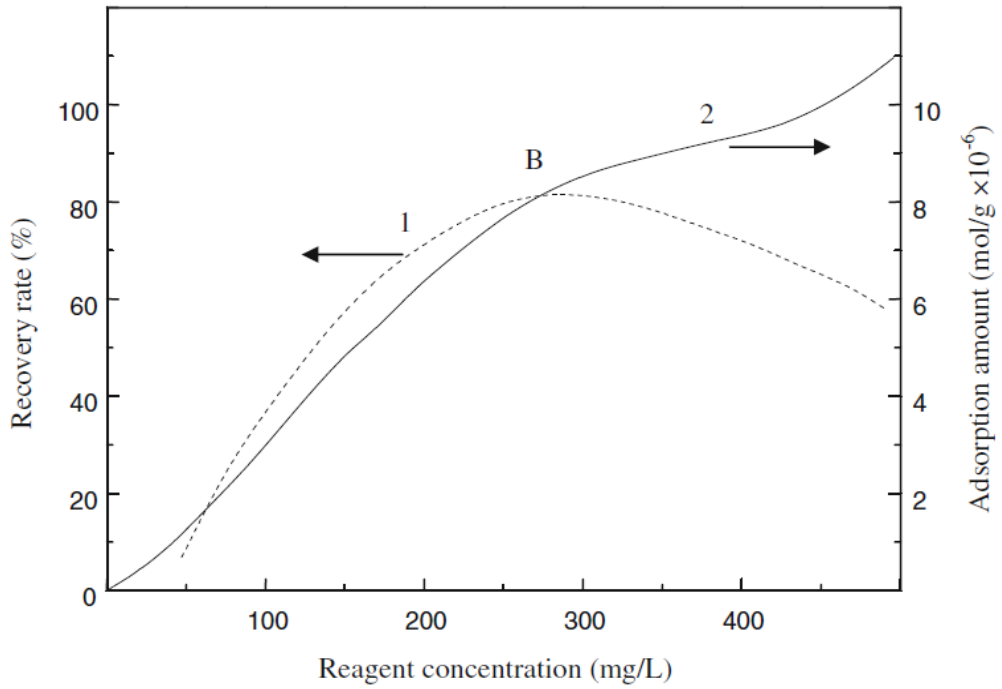

Figure 2.12: The relationship between 8-hydroxyquinoline concentration and adsorption amount and flotation recovery of wolframite, where point B represents a brief plateau phase, adapted from Wang (2016)

In a study investigating the effect of collector adsorption on the flotation of a South African PGM ore, Hadler et al. (2005) found that increasing SIBX concentration, and increasing the conditioning time, resulted in higher concentrate grades. Moreover, the final surface concentration of SIBX adsorbed onto the mineral surfaces was dependent on the initial concentration; for an initial dosage of 25 ppm, the final surface concentration was twice that achieved at an initial dosage of 10 ppm, as indicated by the collector remaining in solution (Figure 2.13). However, increasing the collector

concentration did not affect the recoveries of the valuable minerals. Thus, at low particle hydrophobicities, the particles did not affect the froth stability. As their hydrophobicity increased, froth stability reached a maximum, after which further hydrophobicity led to films being ruptured and the increased drainage of water and gangue minerals from the froth phase, hence the increased grades.

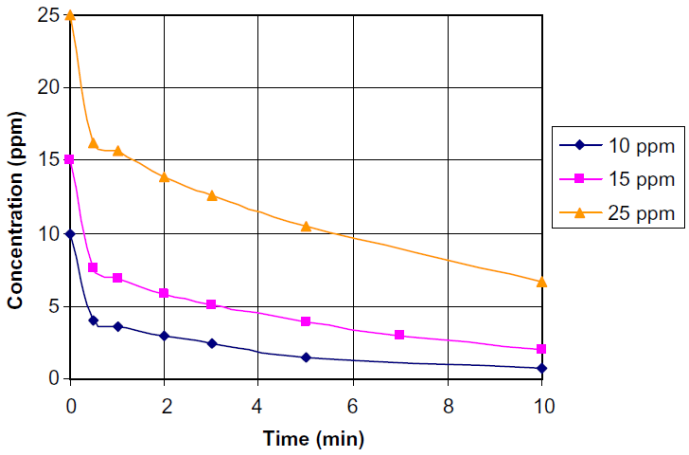

Figure 2.13: The concentration of collector (SIBX) remaining in solution (Hadler et al., 2005)

Finkelstein and Lovell (1972) have demonstrated the rapidity with which collector is abstracted from solution. In a study investigating the effect of Unitol DSR dosage on phoscorite flotation, the collector concentration in solution was reduced from 105 to 8.1 ppm two minutes after the collector was added. They theorised, however, based on a previous study (Lovell and Finkelstein, 1972), that only 4% of the Unitol DSR that had been removed from solution had been adsorbed on phoscorite. It was hypothesised that at the time of sampling and testing, sufficient time for saponification had not passed.

2.4. Flotation of Sulphide Minerals

Because of their differing surface properties, sulphides tend to exhibit varying flotation behaviours even when subjected to the same pulp conditions, as demonstrated by Masiya and Nheta (2014), Plaksin et al. (1954), Senior et al. (1995), Shengo et al. (2016) and many others. Moreover, depending on the mineral, the point at which collector is added to the slurry (be it in the mill or the cell) has been shown to affect mineral recovery. For example, the chalcopyrite recovery of a Merensky ore has been shown to be independent of the point of xanthate addition, while for the same ore, the pentlandite recovery was higher when the collector was added to the mill. This was attributed to the rapid oxidation of newly liberated pentlandite surfaces. As a counter measure, either the collector should be dosed in the mill, or sulphidisation be employed (Wiese et al., 2006).

In a study aimed at selectively floating pentlandite from an ore whose primary gangue mineral was pyrrhotite, Senior et al. (1995) showed that the presence and nature of certain gangue minerals can affect the flotation of sulphide minerals. The recovery-time curves for a slurry containing iron sulphides, pentlandite and talc showed that the initial rates of flotation for pentlandite and iron sulphides were low, while the rate for talc was higher. Moreover, for a collectorless chalcopyrite-talc system, the initial flotation rate for chalcopyrite rapidly decreased with an increase in the amount of talc in the system. In this case, the floatability of one hydrophobic mineral inhibited that of another.

Senior at el. (1995) further observed that the final recovery of nickel was controlled by the coarse and fine fractions of the feed ore. Though the fine and coarse fractions had approximately equal nickel contents, more than half of the unrecovered nickel was in the fine fraction. Increasing the collector dosage resulted in increased pentlandite recoveries, but at the same time, the separation selectivity was reduced as more pyrrhotite was recovered (resulting in lower grades). Additionally, increasing the duration of flotation resulted in the increased recovery of fine pentlandite particles; but again, the selectivity was compromised. The chain length of the xanthate (n-amyl xanthate was substituted for ethyl xanthate) showed no perceivable change on the recovery of the fine size fraction.

Plaksin et al. (1954) propose that due to the heterogenous nature of mineral surfaces, combinations of collectors can be used to optimise flotation behaviour. In comparison with the usage of the single collectors that make up the combination, denser adsorption layers are achieved when combinations are used, resulting in improved mineral recoveries. The hypothesis underlying this improvement is thus: "*The simultaneous usage of two collectors provides for the most complete and fastest formation of the adsorption layer on the mineral surface and gives more efficient flotation; [this is because of] the non-homogeneity of the adsorption behaviour of the mineral surfaces and the presence of individual regions with various activity concerning different collectors*."

2.5. Ore Mineralogy

Two sulphide ores were investigated in this study. The first was a high grade copper ore from the Zambian Copperbelt region, and the second was a low grade nickel-copper ore from Lapland Finland.

2.5.1. Properties of the High Grade Copper Ore

The Central African Copperbelt extends for approximately 450 km from northern Zambia and into the Democratic Republic of the Congo. The Kansanshi ore deposit, where the high grade copper ore pertinent to this study was sourced, is located in the north western province of Zambia, approximately

180 km northwest of the town of Chingola (Broughton, 2014). Figure 2.14 shows the location of the Kansanshi deposit along the Zambian Copperbelt.

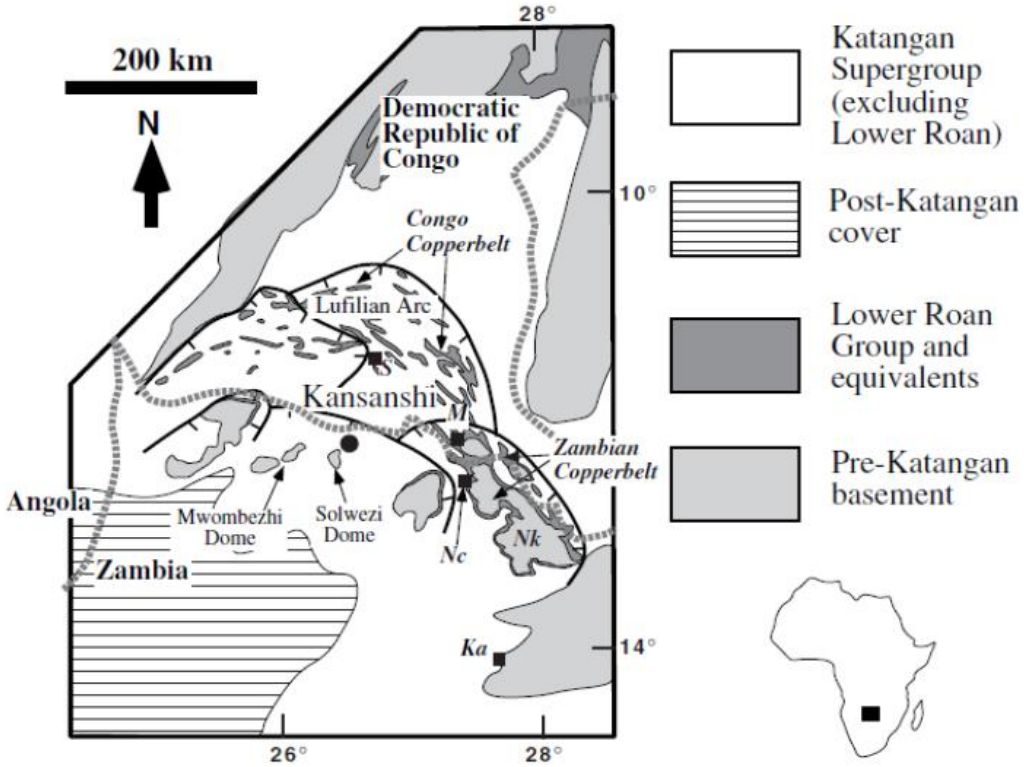

Figure 2.14: Location of the Kansanshi deposit along the Zambian Copperbelt, adapted from Kalichini (2015)

Kansanshi mine processes three types of copper ores: sulphide, oxide and mixed. The ores are classed as either high or low grade; with the high grade ore predominantly comprising sulphides, and the low grade ore comprising a higher proportion of oxides. The quality classification is based on the content of the acid-soluble copper (AsCu) in the ore, such that the AsCu content is a proxy for oxide minerals (Ngulube et al., 2016).

The total copper content of the high grade ore is approximately 1%. The primary copper-bearing mineral in the deposit is chalcopyrite ($CuFeS_2$), with a content of 3.9% by weight fraction; the secondary sulphides are chalcolite (Cu_2S) and bornite (Cu_5FeS_4), digenite (Cu_9S_5) and covellite (CuS). The most prominent gangue minerals for the deposit have been identified as quartz, calcite, dolomite, pyrrhotite and pyrite (Broughton, 2014). Table 2.2 presents a summary of the average bulk mineralogy of the high grade ore used in this study. Mica, plagioclase-feldspar and quartz comprised the bulk gangue minerals. Figure 2.15 illustrates the deportment of copper, as determined by Kalichini (2015).

Table 2.2: The average bulk mineralogy of the high grade copper ore, adapted from Kalichini (2015)

Mineral	Content (wt%)	Mineral	Content (wt%)
Pyrite	3.0	Amphibole	0.7
Pyrrhotite	0.8	Mica	23.3
Chalcopyrite	3.9	Kaolinite	0.3
Bornite	<0.1	Plagioclase-feldspar	32.7
Chalcocite/digenite	<0.1	Quartz	25.4
Covellite	<0.1	Calcite	4.2
Other sulphides	<0.1	Fe-Ti minerals	1.7
Cuprite	<0.1	Limonite	2.7
Malachite/azurite	<0.1	Others	1.1
Chrysocolla	0.1		

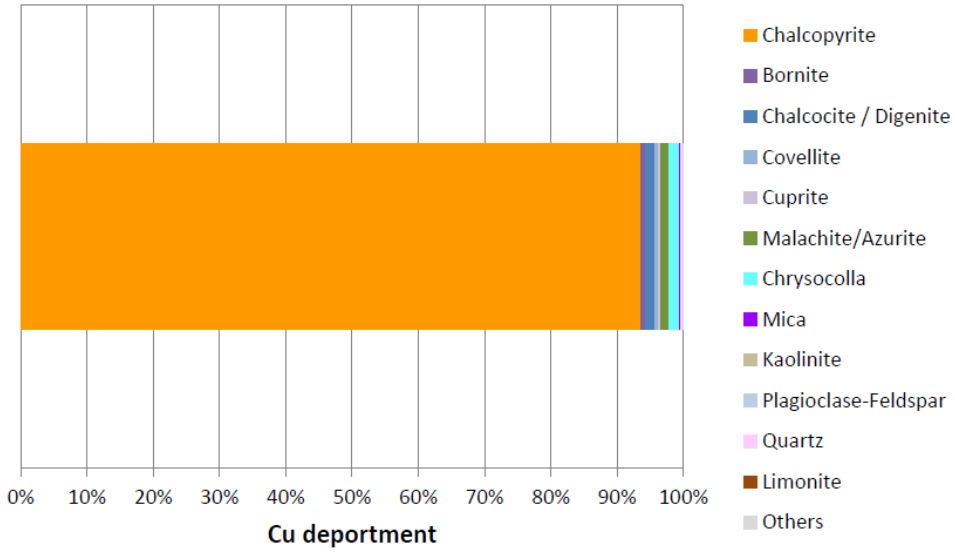

Figure 2.15: The distribution of copper in the high grade copper ore (Kalichini, 2015)

The high grade copper ore will henceforth be referred to as Ore A.

2.5.2. Properties of the Nickel-Copper Ore

The Kevitsa Igneous Complex is located in the Central Lapland Greenstone Belt (CLGB), in Finland, nearly 140 km north of the Arctic Circle. The deposit, where the nickel-copper ore pertinent to this study was sourced, consists of disseminated sulphides. Figure 2.16 shows the location of the complex relative to CLGB. The mineralisation of the deposit is classed into two economically valuable ore types. The first is a regular Ni-Cu ore which, on a weight basis, is more enriched in copper-bearing than nickel bearing minerals. The second and far less abundant, is the Ni-PGE ore, which contains more nickel-bearing than copper-bearing minerals. In both cases, chalcopyrite is the primary copper-bearing mineral, with secondary cubanite. Pentlandite, followed by millerite (with the latter contained

mainly in the Ni-PGE ore), are the nickel-bearing minerals (Gray et al., 2016; Luolavirta et al., 2016). Of concern to this study is the regular Ni-Cu ore.

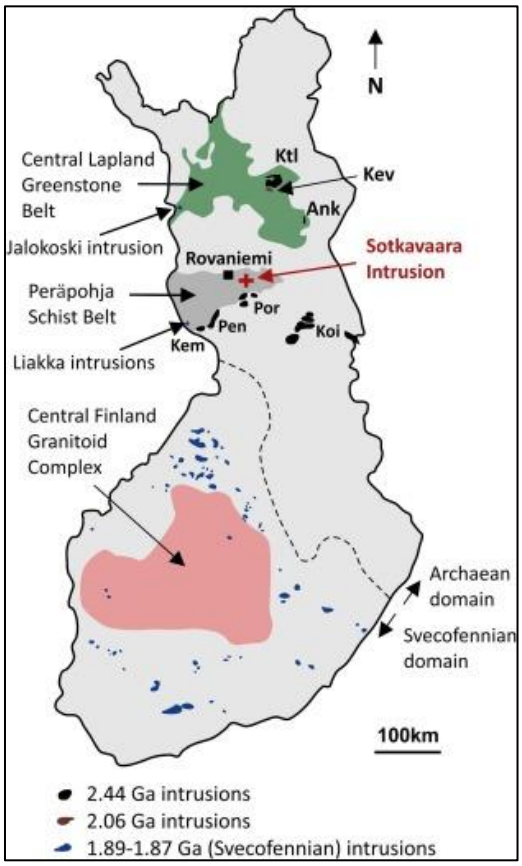

Figure 2.16: The location of the Kevitsa deposit relative to the Central Lapland Greenstone Belt, adapted from Guice et al. (2017)

The sulphides account for less than 5% of the deposit's mineralogy, with up to 20% of the nickel and up to 10% of the copper often contained in non-sulphide minerals, rendering the deposit low grade. The most ubiquitous sulphide gangue mineral is pyrrhotite; additionally, oxidic minerals such as chromite and magnetite are present in the ore. Of the non-sulphide gangue minerals, pyroxene is the most ubiquitous (Muzinda and Schreithofer, 2017a). Table 2.3 is a summary of the estimated reserve as enclosed within New Boliden Kevitsa Oy, and Table 2.4 presents a summary of the average bulk mineralogy of the regular nickel-copper ore, with diopside and amphibole-hornblende comprising the bulk of the gangue minerals.

Table 2.3: The estimated reserve of economically valuable of metals on New Boliden Kevitsa as of 2016, adapted from Muzinda and Schreithofer (2017a)

Metal	Ni	Cu	Au	Pd	Pt
Ubiquity	0.219%	0.335%	0.095 ppm	0.130 ppm	0.200 ppm

Table 2.4: The average bulk mineralogy of the regular nickel-copper ore, (sourced from plant operational logs)

Mineral	Content (wt%)	Mineral	Content (wt%)
Pentlandite	0.316	Chlorite	4.38
Millerite	0.006	Diopside	38.6
Troilite	0.749	Hypersthene	0.489
Pyrrhotite	0.493	Enstatite	2.04
Chalcopyrite	1.18	Quartz	0.06
Cubanite	0.877	Olivine	6.85
Magnetite	1.99	Albite	0.785
Biotite	0.829	Anorthite	1.84
Talc	0.621	Dolomite	0.272
Amphibole-hornblende	33.0	Calcite	0.006
Serpentine	3.79	Other	0.906

Henceforth, the nickel-copper ore will be referred to as Ore B.

2.6. Water Quality and Its Effects on Flotation

Interactions occurring in the pulp phase are not only affected by the components intended to be there (such as the reagents and minerals), but by any impurities present. The stringency of fresh water usage in concentrator plants, as well as other water-recycling practices result, often, in a chemically altered pulp phase. Recycled water contains more dissolved solids and thus a higher ionic strength, all of which could alter the mineral properties and thereby alter the reactions occurring at the mineral surfaces (Bailey, 1970; Levay et al., 2001; Pickett and Joe, 1974; Slatter et al., 2009).

The ionic strength of a solution is defined as the measure of the concentration of electrolytes dissolved in that solution. Recycled water is usually obtained from tailings dams (and occasionally, from the thickener overflow), where the typical contaminating species are: SO_4^{2-}, Cl^-, F^-, Mg^{2+}, Ca^{2+}, Na^+, K^+, base metals, collectors, frothers, activators, depressants, colloidal materials such as iron hydroxides, and natural organic material. Additionally, some ions exist naturally in water (Sinche-Gonzalez et al., 2016; Slatter et al., 2009).

Empirical findings suggest that species dissolved in solutions strongly influence collector adsorption and overall flotation response (Fuerstenau and Chander, 1986; Leja and Schulman, 1954). This is because some of the aforementioned chemical species have been demonstrated to be surface active on oppositely charged mineral surfaces (Hadler et al., 2005; Yoon and Basilio, 1993). For example, in a system where oleate was used as a collector, when the concentration of the pH regulator (Ca^{2+}) in

solution exceeded the concentration of oleate, the collector precipitated in the bulk solution and depression of the valuable mineral was observed (Fuerstenau and Chander, 1986).

Yet there are still advantages to recycling process water, some of which are: the reduced freshwater use and discharge from the plant, and the retention of some reagents (especially for thickener overflows), therefore lowering the consumption of reagents (Slatter et al., 2009). However, the retention of residual reagents can be a disadvantage, as in the following case: the recirculation of frother can cause overly stable froths, which can lead to reduced selectivity and lowered concentrate grades (Backx, 2016; Picket and Joe, 1974). Other drawbacks are as follows (Bruckard et al., 2011; Levay et al., 2001; Shengo et al., 2016; Sinche-Gonzalez et al., 2016; Slatter et al., 2009):

- Reduced reagent efficiency as a result of increased suspended solids in the recycled water.
- Increased pollutant levels such as chemical and microbiological oxidation in the water.
- Loss of recovery due to: competitive adsorption between reagents and other dissolved species; loss of reagents due to precipitation; over-coating, and unwanted reactions.
- Loss of grade due to: non-selective coating of reagents on valuable and gangue minerals, and ineffective separation of gangue minerals.
- The hydrophobic coating of microbiological material on commercially valueless minerals, causing unwanted flotation.
- Decreased hydrophobicity of valuable mineral surfaces due to the adsorption of hydrophilic groups in organic materials. This in turn hinders bubble-particle adhesion.

2.6.1. Effects of Ionic Strength on the Pulp Phase

Calcium, iron and sulphate ions are among the most common species dissolved in sulphide process waters. The calcium ions often originate from the addition of lime, which is commonly used to maintain the pH of the pulp phase. The sulphate ions often originate from the addition of reagents used to control the surface properties of the minerals (Kirjavainen et al., 2002). Depending on the grinding environment, pulp potential and other factors, these ions have been shown to either increase or decrease the floatability of sulphide minerals (Ikumapayi et al., 2012; Slatter et al. (2009).

In a study by Corin et al. (2011), in which the effect of ionic strength on a nickel-copper ore was investigated, increasing the ionic strength (per Table 2.5) had no apparent effect on the mineral recoveries. Additionally, increasing the ionic strength resulted in higher water recoveries to the

concentrate; and with that, slight decreases in entrained and naturally floating gangue material per unit water mass. The latter result was attributed to the possible adsorption of calcium and magnesium ions on the gangue surfaces, hence reducing their hydrophobicity. However, higher ionic strengths led to reduced concentrate grades because the final mass pulls and water recoveries increased and thus the total gangue recovery increased with increasing ionic strength.

The above-discussed complemented a study by Kirjavainen et al. (2002), in which the effect of calcium and thiosulphate ions on a nickel-copper sulphide ore was investigated. In this case, the calcium ions activated the xanthate adsorption when the galvanic effect of mill iron was effective. But the thiosulphate ions indirectly increased mineral floatability by decreasing the adsorption of hydrophilic compounds (such as metal hydroxides) on the mineral particles, resulting in increased copper and nickel recovery.

However, investigating the flotation of pure sulphide minerals (galena and chalcopyrite) under the influence of increasing calcium and sulphate concentrations, Ikumapayi et al. (2012) found that the ions reduced mineral recoveries at concentrations up to 50 and 100 mg/L respectively, beyond which there were no observed changes. In the presence of Ca^{2+}, the reduced floatability was attributed to the formation of hydrophilic layers of $CaCO_3$ on the minerals. This inhibited the xanthate adsorption, as shown by spectrometric analyses. The sulphate ions were also shown to inhibit collector adsorption. However, when the minerals were floated in process water (as opposed to water containing only one ion species), an increase in recovery was observed. This was attributed to the presence of residual reagents that enhanced xanthate adsorption.

The iron ions in process water result because most sulphide minerals are more electrochemically noble than the media used to grind them. As such, a galvanic couple exists between the media and mineral, which increases media corrosion rate (Greet et al., 2004; Subrahmanyan and Forssberg, 1995). The corrosion debris precipitate onto the minerals and form metal hydroxyl complexes, which impair collector adsorption (Guo and Yen, 2005). For chalcopyrite, the metal-xanthate species responsible for flotation is stable such that the hydrophilic hydroxyl ions only depress the mineral at pH values above 13. For pyrite, pH adjustment by adding lime leads to pyrite depression above a pH of 6.9, when the mineral surface charge is negative and the calcium ions are adsorbed by electrostatic attraction (Fuerstenau, 1982c).

2.6.2. Effects of Ionic Strength on the Froth Phase

The electrolytes of some salts have been shown to, above a given concentration (approximately 0.1 M), retard bubble coalescence (Craig, 2004). Moreover, empirical studies have demonstrated that process waters with higher ionic strengths improve flotation by causing very stable froths, and that some of the dissolved ions have a lesser effect on flotation than others. For example, sulphate has been demonstrated to have a more negative effect than chloride. The stabilised froth leads to a higher water content in the froth phase, which tends to result in reduced selectivity. But overall, using recycled water has been reported to decrease reagent consumption by approximately 50% (Slatter et al., 2009).

In order to investigate the effects of ionic strength on the froth phase, a study by Manono et al. (2013) distinguished between froth, which comprises three phases: water, air and mineral particles, and foam, which comprises only two phases: water and air. The reagents and electrolytes dissolved in the water were common for both systems, with the solution being considered a single phase, thus foam stability was likened to froth stability.

The foam collapse rate, which is a function of the foam collapse time, indicates the froth stability; this is because unstable froths are prone to high collapse rates; this in turn negatively affects bubble–particle attachment, which may result in a loss of recovery (Lovell, 1982; Taggart and Hassialis, 1946). Hence, the key foam stability indicators per Manono et al. (2013) were foam height and collapse time, and bubble size.

The findings showed that with increasing ionic strength in the presence of a low frother concentration, the water recovery to the concentrate also increased. In addition, increasing the ionic strength resulted in reduced bubble sizes due to reduced bubble coalescence; the foam height and foam collapse time were also increased, indicating a more stable foam. The chosen frother was a non-ionic polyglycol ether, as such, it was not expected to interact with the ions. It was therefore concluded that the ions did not improve the frother's performance but instead, enhanced foam stability alongside the frother (Manono et al., 2013). Similar findings were observed by Corin et al. (2011) and Shengo et al. (2016).

2.6.3. A Chemical Summary of the Relevant Synthetic Plant Waters

The University of Cape Town Centre for Minerals Research has developed a standard synthetic plant water formula (with a total dissolved concentration of 1023 mg/L, ionic strength of 0.0213 M and termed 1SPW) that once reflected the typical water analysis of a number of South African PGM concentrators. This is achieved by adding various chemical salts to distilled water (Wiese et al, 2005).

In comparison, Cape Town tap water has a TDS of 68.5 mg/L. Recent concentrator analyses show an increase in the amount of dissolved ions in recycled water. As such, standard plant water concentrations are no longer typical of most platinum group element (PGM) concentrators. The influence of ionic strength on flotation is thus studied by tripling and multiplying five times, the level of ions in the standard plant water, as demonstrated in Table 2.5. To avoid precipitation, the salts are added to the water in the order indicated in Section 4.3 (Corin et al., 2011; Wiese et al., 2005).

Table 2.5: The concentrations of the ions present in synthetic plant water (Wiese et al., 2005)

Water Type	Ca^{2+} (ppm)	Mg^{2+} (ppm)	Na^{+} (ppm)	Cl^{-} (ppm)	SO_4^{2-} (ppm)	NO_3^{-} (ppm)	CO_3^{2-} (ppm)	TDS (mg/L)	Ionic strength (M)
Tap	19	1.5	12	12	19	17	-	68.5	0.0020
1SPW	80	70	153	287	240	176	17	1023	0.0213
3SPW	240	210	459	861	720	528	51	3069	0.0620
5SPW	400	350	765	1435	1200	880	85	5115	0.0977

3. OBJECTIVES, HYPOTHESIS AND KEY QUESTIONS

3.1. Problem Statement

Environmental concerns, especially for water-scarce regions like South Africa, have resulted in the need to recycle process water within mining plants and hence limit the usage of potable water for industrial applications. Given the difference in the chemical composition of potable and recycled water, it is essential to understand the effects of water quality on mining processes, especially flotation—a process that is strongly affected by varying system conditions. Previous studies have demonstrated the dependence of the efficiency of flotation separation on valuable mineral hydrophobicity, and that the hydrophobicity is in turn dependent on the degree of adsorption of collectors. It is therefore essential to understand the effects of water quality on collector adsorption and the resultant flotation response.

3.2. Objectives of the Study

The objectives of the study are therefore:

- To investigate the effect of collector adsorption and distribution on the flotation performance of a high grade copper ore, in the presence of increasing ionic strength.
- To determine the number of recycles necessary to render the ions and collector dormant; this, at varying concentrations of each.
- To investigate the effect of collector adsorption and distribution on the flotation performance of a nickel-copper ore in the presence of different plant waters.

3.3. Statement of Key Questions

1. How does increasing plant water ionic strength affect the collector adsorption in the flotation of a Kansanshi copper ore and a Lapland nickel-copper ore?
2. How does increasing plant water ionic strength affect the collector distribution in the flotation of a Kansanshi copper ore and a Lapland nickel-copper ore?
3. How does initial collector dosage affect the flotation response of a Kansanshi copper ore? What are the adsorption profiles of varying initial collector dosages?

4. How does recycling the collector retained in plant water affect the flotation response of a Kansanshi high grade copper ore and a Lapland nickel-copper ore?

5. Which, between the collector dosage and water quality, has a greater effect on the flotation response of a Kansanshi high grade copper ore and a Lapland nickel-copper ore? Can their respective effects be decoupled?

3.4. Research Hypothesis

The hypothesis is as follows: increased plant water ionic strength has been shown to possess froth-stabilising abilities; low collector dosages have been shown to have negligible effect on the froth phase, but at higher concentrations they destabilise the froth, hence:

- At low ionic strengths, increasing the collector dosage will result in increased valuable mineral recovery and concentrate grades; this, due to increasing mineral hydrophobicity and froth destabilisation.

- At high ionic strengths, increasing the collector dosage will result in high valuable mineral recovery but poor concentrate grades; this, because plant water ionic strength has a greater effect on the froth phase than the collector.

3.5. Sustainability Aspects of the Research

The study supplements the following: the growing knowledge on recycling process water within mining operations; the reduced usage of freshwater (and hence availing this water for domestic, agricultural and other such applications), and the reduced discharge of process waters to the environment. Some of these waters contain toxic contaminants which might pollute open and ground water reserves, which may in turn harm the flora and fauna dependent on these water sources.

The study is therefore related to Sustainable Development Goals 6, 9, 12 and 15. The respective mission statements of these are: to ensure water and sanitation for all; to build resilient infrastructure, promote sustainable industrialization and foster innovation; to ensure sustainable consumption and production patterns; and to conserve and sustainably use the oceans, seas and marine resources (The United Nations, 2015).

4. EXPERIMENTAL DETAILS

4.1. Ore Sampling and Preparation

The high grade copper ore—Ore A, was first crushed to a particle size of -8 mm and homogenised. The ore was then split into 1 kg portions which were sealed in plastic bags and sent to UCT. As confirmed by XRF and ICP-OES analysis, the copper and iron content of the samples were consistent. The average tested copper and iron content were 1.14 and 3.72 wt.%, respectively.

The Finnish nickel-copper ore—Ore B, was also crushed to a particle size of -8 mm and homogenised. The bulk sample was dried at 83°C, after which it was homogenised by using a rotary riffle splitter. The samples were then weighed into 1 kg portions. As confirmed by XRF analysis, the copper, iron and nickel content of the ore were consistent throughout the tested samples. The average tested copper, nickel and iron content were 0.359, 0.249 and 5.872 wt.%, respectively.

4.2. Lab-Scale Milling

A stainless steel rod mill was used for comminution of Ore A. The dimensions of the mill and the operational parameters are shown in Table 4.1.

Table 4.1: Mill characteristics as employed for Ore A

Parameter	Value
Mill diameter (mm)	200
Mill depth (mm)	297
Number of rods (-)	20
Diameter of rods (mm)	6×25, 8×20 and 6×16
Mill speed (rpm)	256
Slurry residence time (minutes)	13.55

For each test, a 1 kg sample of ore and 500 ml of the relevant synthetic plant water were added to the mill. For example, when the flotation response of the ore was tested at 1SPW, 1SPW was used in the milling process; the same was done for 3SPW and 5SPW. The residence time and mill drive speed achieved a particle size distribution of 60% passing 75 μm, per mill calibration (Wiese et al., 2005).

A stainless steel rod mill was used to mill Ore B. The dimensions of the mill and the operational parameters are shown in Table 4.2.

Table 4.2: Mill characteristics as employed for Ore B

Parameter	Value
Mill diameter (mm)	220
Mill depth (mm)	265
Number of rods (-)	16
Diameter of rods (mm)	23
Mill speed (rpm)	47
Slurry residence time (minutes)	30

For each test, a 1 kg sample of ore and 630 ml of the relevant water type were added into the mill. The residence time and mill drive speed, as shown in Table 4.2, achieved a particle size distribution of 70% passing 75 μm, as per mill calibration.

4.3. Plant Water Preparation

The synthetic plant waters were prepared by adding varying amounts of inorganic salts to distilled water, as shown in Table 4.3 (for 40 litres of 1SPW). To avoid precipitation, the salts were added to water in the order in which they are provided in Table 4.3, i.e. magnesium sulphate was added first, followed by magnesium nitrate, then calcium nitrate, etc. In order to synthesise 3SPW and 5SPW, the concentrations of the respective ions were tripled and multiplied five times by adding three and five times the salt content, respectively (Wiese, 2009). All salts were supplied in powder-form by Merck.

Table 4.3: Inorganic salts used in the synthetic plant water recipe

Inorganic salt	Chemical formula	Mass of salt in 40 L 1SPW (g)
Magnesium sulphate	$MgSO_4.7H_2O$	24.6
Magnesium nitrate	$Mg(NO_3)_2.6H_2O$	4.28
Calcium nitrate	$Ca(NO_3)_2.4H_2O$	9.44
Calcium chloride	$CaCl_2$	5.88
Sodium chloride	NaCl	14.2
Sodium carbonate	Na_2CO_3	1.20

The resultant ionic strengths of the synthesised waters are as reported in Table 2.5 (Section 2.6). In addition to the synthetic plant waters, the flotation response of Ore B was tested in actual plant waters obtained on site, which were as follows:

- Standard process water.
- The copper thickener overflow and the nickel thickener overflow.

- Mine pit waters.
- Raw water, which was sourced from a nearby river.

4.3.1. A Chemical Summary of the Actual Plant Waters

The plant waters were sourced from a mine that recycles and reuses approximately 90 to 95% of its processing water (Muzinda and Schreithofer, 2017b). The water balance is illustrated in Figure 4.1.

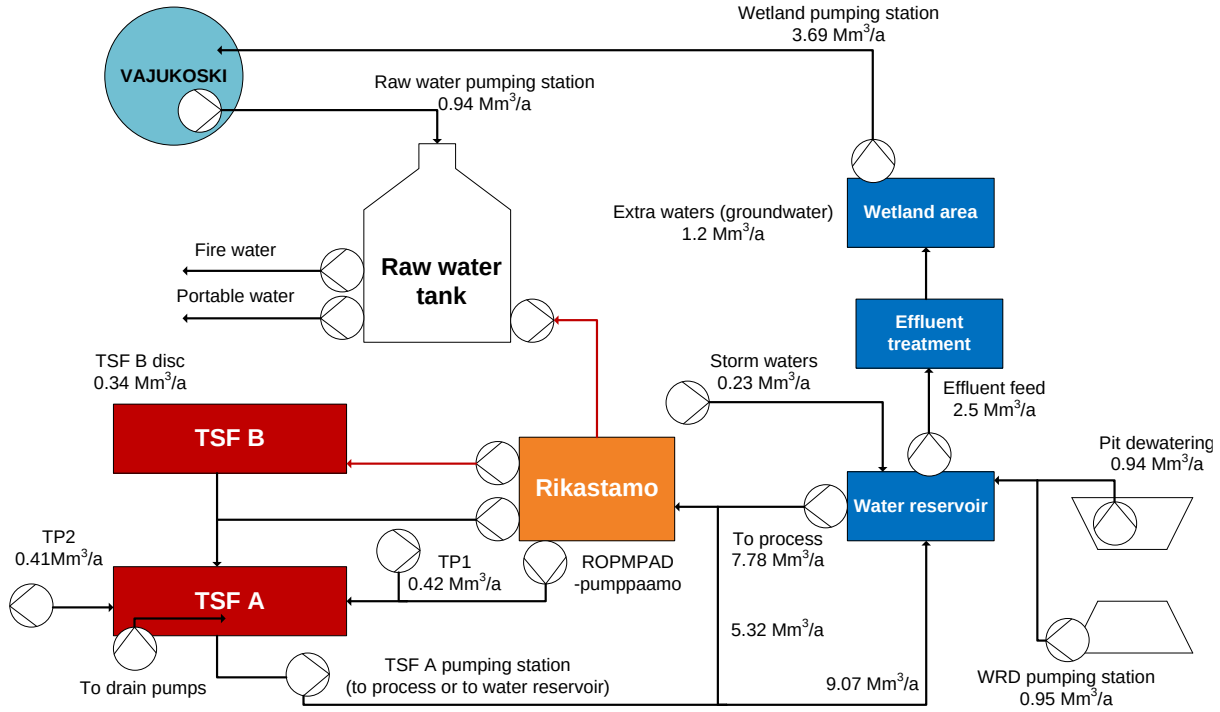

Figure 4.1: The water balance for Kevitsa Mining Oy, as of 2015, adapted from Muzinda and Schreithofer (2017a)

The location of the mine (140 km north of the Arctic Circle) results in vastly different seasonal conditions, and these have been observed to strongly affect the quality of the process water. The quality, as measured by conductivity—which is linked to the ionic strength via Equation 4.1, is notably poorer in the winter months. This is because in summer, UV light from the abundant sunlight hastens the decomposition of organic materials such as the xanthate collector; while in winter, with fewer hours of sunlight, much of the residual xanthate is recycled before decomposing. Moreover, in winter an ice cap forms over the tailings pond, further hindering the decomposition of dissolved species (Backx, 2016; Muzinda and Schreithofer, 2017b).

$$I = 1.6 \times 10^{-5} \cdot EC \qquad 4.1$$

In Equation 4.1, $\boldsymbol{I}$ denotes the ionic strength and $\boldsymbol{EC}$ the electrical conductivity (Aqion, 2015).

The water quality variations are compounded by the polymetallic nature of the ore (Ore B) and sequential configuration of the flotation circuit. The plant produces two concentrates—with the copper concentrate floated first, followed by the nickel. The tailings from the nickel circuit are sent to a third circuit, where remaining sulphides are removed to produce low-sulphur tailings with no acid generating potential (Gray et al., 2016). Reagents used in each circuit are summarised in Table 4.4.

Table 4.4: Reagents and conditions in the flotation circuit, adapted from Gray et al. (2016)

Reagent/conditions	Cu Flotation	Ni Flotation	Sulphur Flotation
Collector	Aerophine 3418A	SIPX	PIAX
Frother	Nasfroth 240	Nasfroth 240	Nasfroth 350
Depressant	Carboxy methyl cellulose	Triethylene tetramine	-
pH modifier	Lime	Sulphuric acid	Provisional
pH	11-12	<11	Provisional

Recycled xanthate complicates the process in the following way: due to the fast flotation kinetics of chalcopyrite and the slow kinetics of pentlandite, a weaker collector is used in the copper circuit. In the case of increased recycled xanthate—which is a stronger collector, nickel is floated in the copper circuit (Gray et al., 2016). Kevitsa has established a parameter—known as water intensity—which relates mineral recovery, reagent consumption and water recycling. As can be deduced from Equation 4.2, less fresh water usage and more recycling results in lower water intensities. In addition to the factors already discussed, the winter months are marked by low water intensity because cooling water for mill motors is in low demand. Recoveries for both nickel and copper are reduced during high intensity periods (Muzinda and Schreithofer, 2017b).

$$\text{Water intensity} = \frac{\text{Fresh water intake for the day}}{\text{Milled tons for the day}} \qquad 4.2$$

The calibration model presently used for the determination of residual xanthate in recycled water is as shown in Equation 4.3, where $\boldsymbol{y}$ is the collector concentration in mol/L, and the absorbance is measured by UV/Vis spectrometry (Muzinda and Schreithofer, 2017b).

$$y = 9.5907 \times \text{measured absorbance} \qquad 4.3$$

First, the water sample is filtered to remove suspended solids. Two xanthate absorbance measurements are then taken. The first measurement, A1, is the absorbance of the sample at pH 6.5-6.8. The sample pH is then adjusted to 2 where, in theory, all xanthate is decomposed. The second measurement, A2, is then taken after the sample pH is adjusted back to 6.5-6.8, and the obtained

absorbance peak is a result of concentrations of other suspended chemical compounds. As such, the measure allocated to the xanthate absorbance is taken as the difference between A1 and A2 (Jones and Woodcock, 1973). However, on-site measurements for Kevitsa suggest that the pH-modification method is superfluous, as incomplete xanthate decomposition has been observed, especially at higher xanthate concentrations (Muzinda and Schreithofer, 2017b).

Other commonly accumulated species in Kevitsa process water are as follows: Ca^{2+}, Na^{+}, Mg^{2+}, K^{+} and SO_4^{2-} and frother. Presently, the highest water conductivity is observed in late winter (February and March), followed by a stark decrease in late spring. But overall, there is a rise in the conductivity as a result of the gradual accumulation of ionic species (Muzinda and Schreithofer, 2017b).

Experimental tests for this study were conducted in August 2017. During this time, the spring melt had released water from the ice cap, and raw water intake for mill motor cooling had increased. As a result, the xanthate concentration in the process water recycled from the tailings pond was not significant. Table 4.5 is a summary of chemical properties for the relevant Kevitsa plant waters. The waters with the lowest TDS values also had the lowest conductivities.

Table 4.5: Ionic strengths of the actual plant waters, adapted from Backx (2016); sourced from plant logs

Water type	Process	Cu thickener	Ni thickener	Mine pit	Raw
Conductivity (µS/cm)	1700	1620	950	540	25
Ionic strength (M)	0.0272	0.0259	0.0152	0.00864	0.0004
pH	7.08	11-12	8.2	7.8	8.5

Figure 4.2 illustrates the ionic strengths of all the tested waters relative to each other, in order of increasing ionic strength.

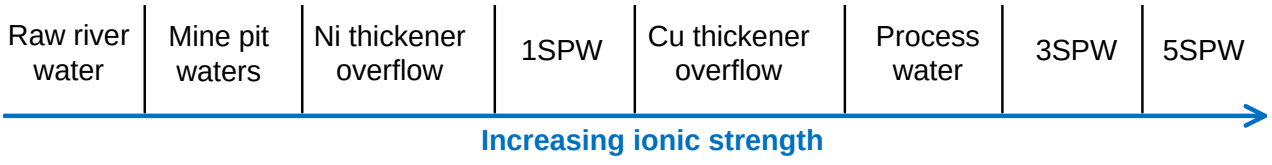

Figure 4.2: The relative qualities of the tested water, as defined by ionic strength

4.4. Reagent Preparation and Storage

The SIPX (in powder form), aerophine (in liquid form) and Nasfroth 240 (in liquid form) were the same as those used for normal Kevitsa plant operations. The SIBX (in powder form) and DOW 200 (in liquid form) were supplied by Senmin and Betachem, respectively.

4.4.1. Xanthate Collector

Both the xanthate collectors (SIBX and SIPX) decompose quickly, thus they were both prepared daily, immediately prior to the experiments (Hadler et al., 2005). 1% collector solutions were prepared as follows: 1 g powder of either SIBX or SIPX was weighed, following which it was dissolved in 100 ml distilled water and sealed (Corin et al., 2011). Residual xanthate solutions were discarded as outlined by the material safety data sheet and laboratory rules.

SIBX was used in the flotation of Ore A, while SIPX was used in the flotation of Ore B. In addition to SIPX, sodium dithiophosphinate (commonly known as aerophine) was used in the flotation of Ore B. The aerophine was dosed as supplied, without dilution, at 5 g/t per run. After use, it was stored at 9°C. Table 4.6 summarises the properties of the collectors.

Table 4.6: Chemical properties of the used collectors

Collector	Chemical formula	Molecular weight (g/mol)	Purity (%)
SIBX	$(CH_3)_2CHCH_2OCS_2Na$	172	97
SIPX	$(CH_3)_2CHOCS_2Na$	158	97
Aerophine 3418A	$(C_4H_9)_2PS_2Na$	232	50

4.4.2. Frother

DOW 200 was used for the flotation of Ore A, and Nasfroth 240 was used for Ore B. Both polyglycol frothers were dosed as supplied, at 50 g/t each. After use, they were stored at room temperature. Table 4.7 summarises the chemical properties of the frothers.

Table 4.7: Chemical properties of the used frothers

Frother	Chemical formula	Molecular weight (g/mol)	Purity (%)
DOW 200	$CH_3(C_3H_6O)_3OH$	206	100
Nasfroth 240	$C_4H_9(OC_2H_4)_3OH$	206	100

4.5. Experimental Method

Two sets of experiments were conducted; the first set was aimed at testing the amount of collector adsorbed onto the mineral particles at different time intervals of the flotation process. The second set was aimed at testing how the collector adsorption affected the respective flotation responses of the ores. This was done at different water types, ionic strengths and initial collector concentrations. The experimental design is illustrated by the factorial designs shown in Figure 4.3 and Figure 4.4.

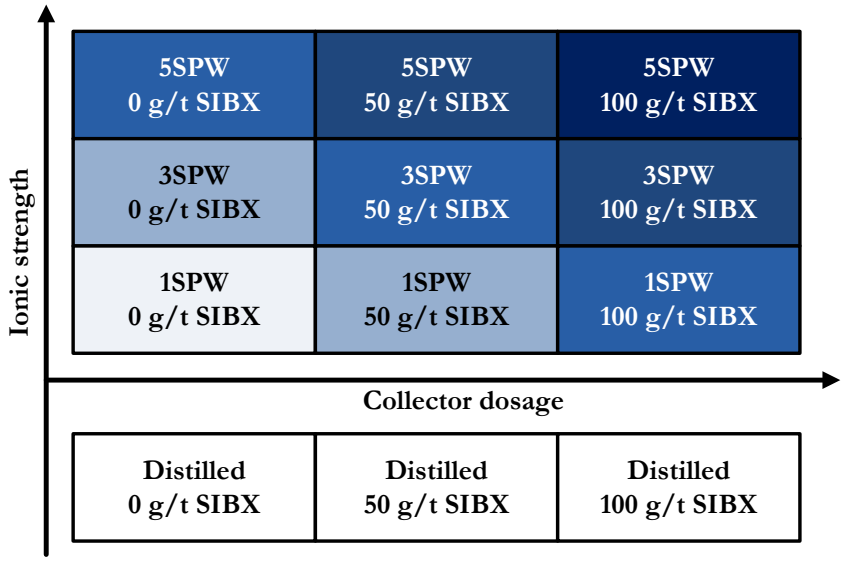

Figure 4.3: A factorial design of the experimental conditions investigated for Ore A

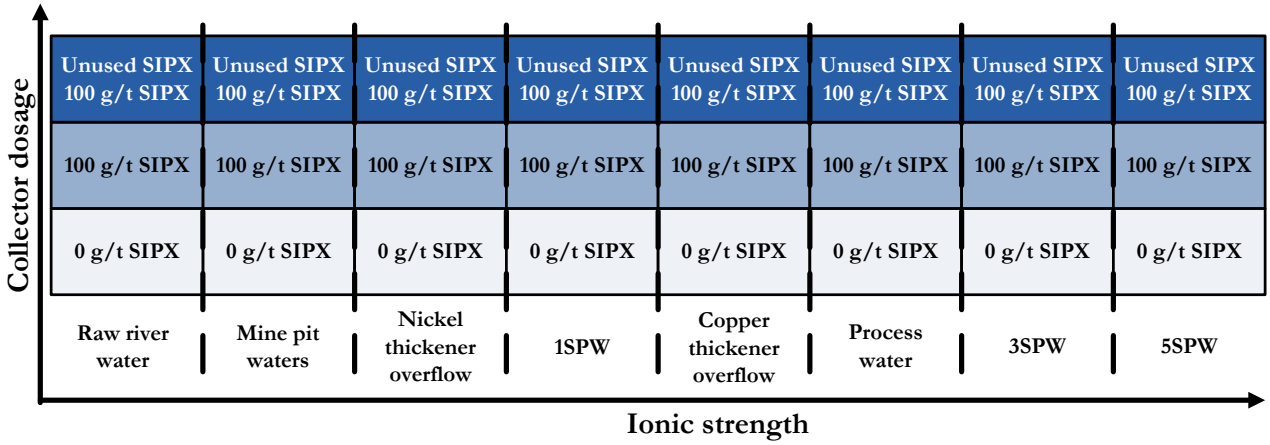

Figure 4.4: A factorial design of the experimental conditions investigated for Ore B

4.5.1. Batch Flotation Tests

For Ore A, the milled slurry was transferred to a 3-litre Barker flotation cell (pictured in Figure 4.5), where the relevant water was added to achieve 35% solids. The cell's speed drive was set to 1200 rpm. A feed sample was taken from the cell, and the slurry was conditioned with either 50 or 100 g/t SIBX and 50 g/t DOW 200 for 2 minutes and 1 minute, respectively. Then the air valve was opened and the air flow rate maintained at 7 L/min for all tests.

At 2, 6, 12 and 20 minutes of flotation, the froth was scraped into a collecting pan every 15 seconds. The froth height (of 2 cm) and water level were maintained by adding the relevant water type (Wiese et al., 2005). Figure 4.6 summarises the experimental scheme followed in floating Ore A.

Figure 4.5: The Barker flotation cell used for all Ore A experiments

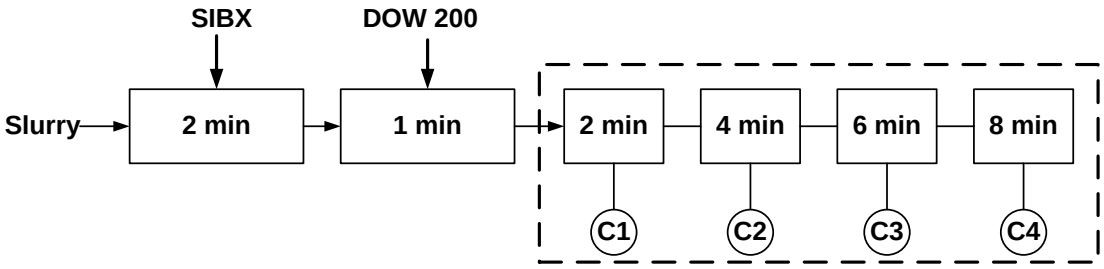

Figure 4.6: A summary of the experimental sequence adopted for the flotation of Ore A

For Ore B, the milled slurry was transferred to a 2-litre flotation cell (pictured in Figure 4.7 along with the mill and UV/Vis spectrophotometer), where the relevant water was added to achieve 34.5% solids. The cell's speed drive was set to 900 rpm. The slurry was conditioned with 5 g/t aerophine and 50 g/t Nasfroth 240 for 1.5 minutes and 30 seconds, respectively. Then the air valve was opened and the air flow rate maintained at 7 L/min for all tests. Aerophine was used specifically to target chalcopyrite since, though not as strong as SIPX, the former is more selective and therefore better suited for the recovery of the fast-floating chalcopyrite (Muzinda and Schreithofer, 2017a).

(a) The Outotec batch flotation cell

(b) The steel grinding mill

(c) The UV/Vis spectrophotometer

Figure 4.7: The experimental setup for Ore B

After 9 minutes of flotation the ore was conditioned with 100 g/t SIPX and additional 50 g/t frother for 1.5 minutes and 30 seconds, respectively. At 1, 3 and 9 minutes of flotation the froth was scraped into a collecting pan every ten seconds to recover copper; the same was done at 11, 15, 21 and 29 minutes to recover nickel. These conditions, the sequential addition of collectors and the conditioning times, were selected in order to mimic plant operational conditions. The water level and froth height (of 2 cm) in the flotation cell were maintained by adding the relevant water type (Hadler et al., 2005; Wiese et al., 2005). Figure 4.8 is a summary of the scheme followed in floating Ore B.

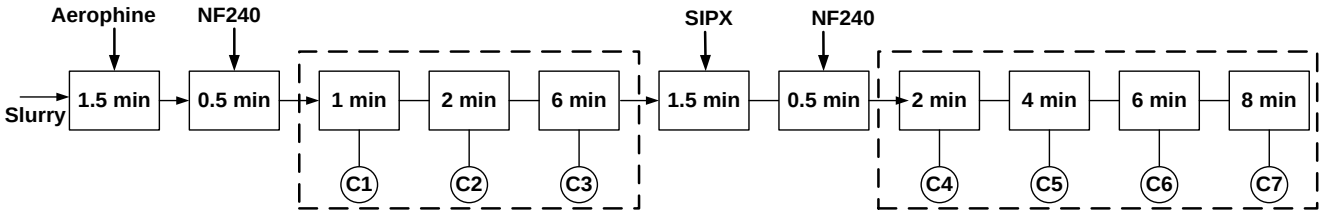

Figure 4.8: A summary of the experimental sequence adopted for the flotation of Ore B

For both ores, the concentrates, feed and tailings samples were filtered, dried and weighed. They were then sent for analysis to determine the amount of copper and nickel contained in each. XRF analysis was used for Ore B while XRF and ICP-OES were used for Ore A.

4.5.2. Simulating Recycle Runs

After the flotation of each ore, the tailings filter solution was used to simulate a recycle run. For Ore A, 62.1% (1.8 litres of 2.9 litres) of the water used at the start of the second run was obtained from a preceding run. For Ore B, 73.7% (1.4 litres out of 1.9 litres) of the water used at the start of the second run was obtained from a preceding run. Figure 4.9 shows the procedure followed in simulating the recycle runs in the flotation of Ore A, while Figure 4.10 shows a similar procedure for Ore B.

All tests were performed in duplicate to allow subsequent error analysis and the examination of reproducibility.

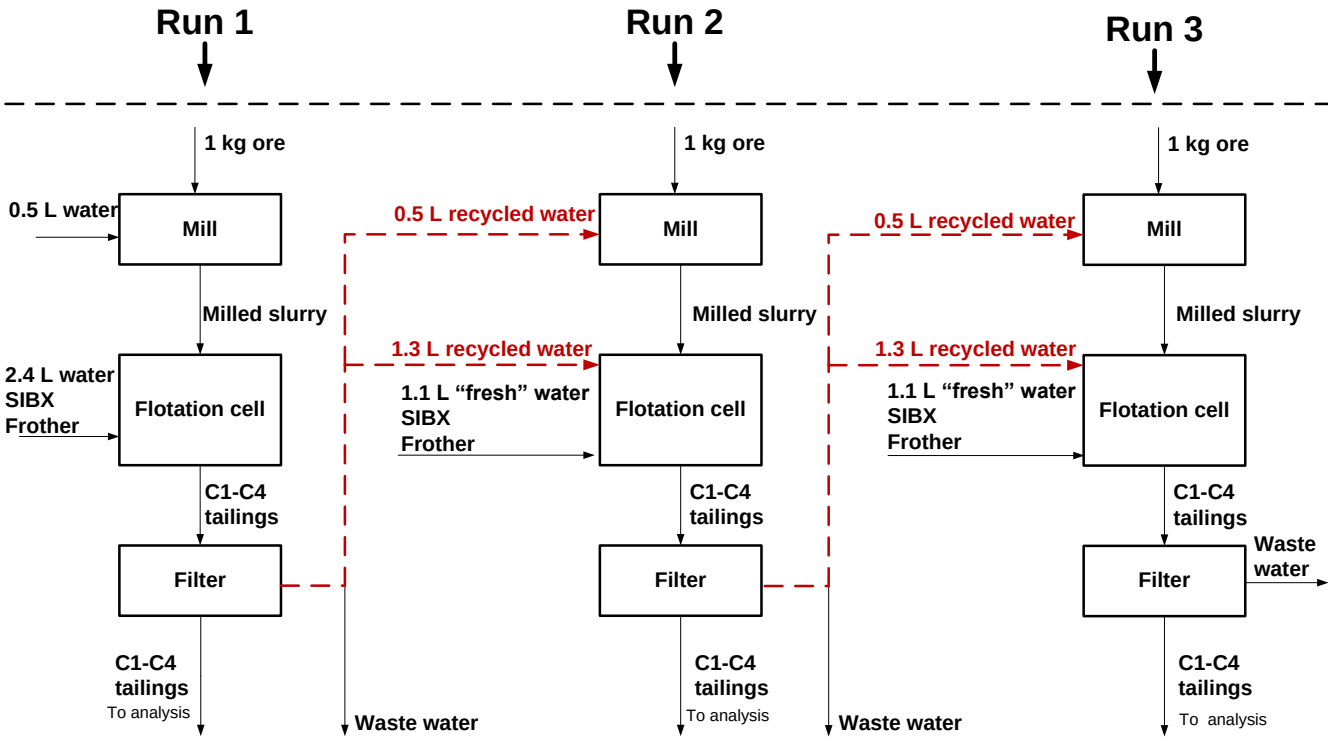

Figure 4.9: the procedure followed in simulating the recycle runs in the flotation of Ore A

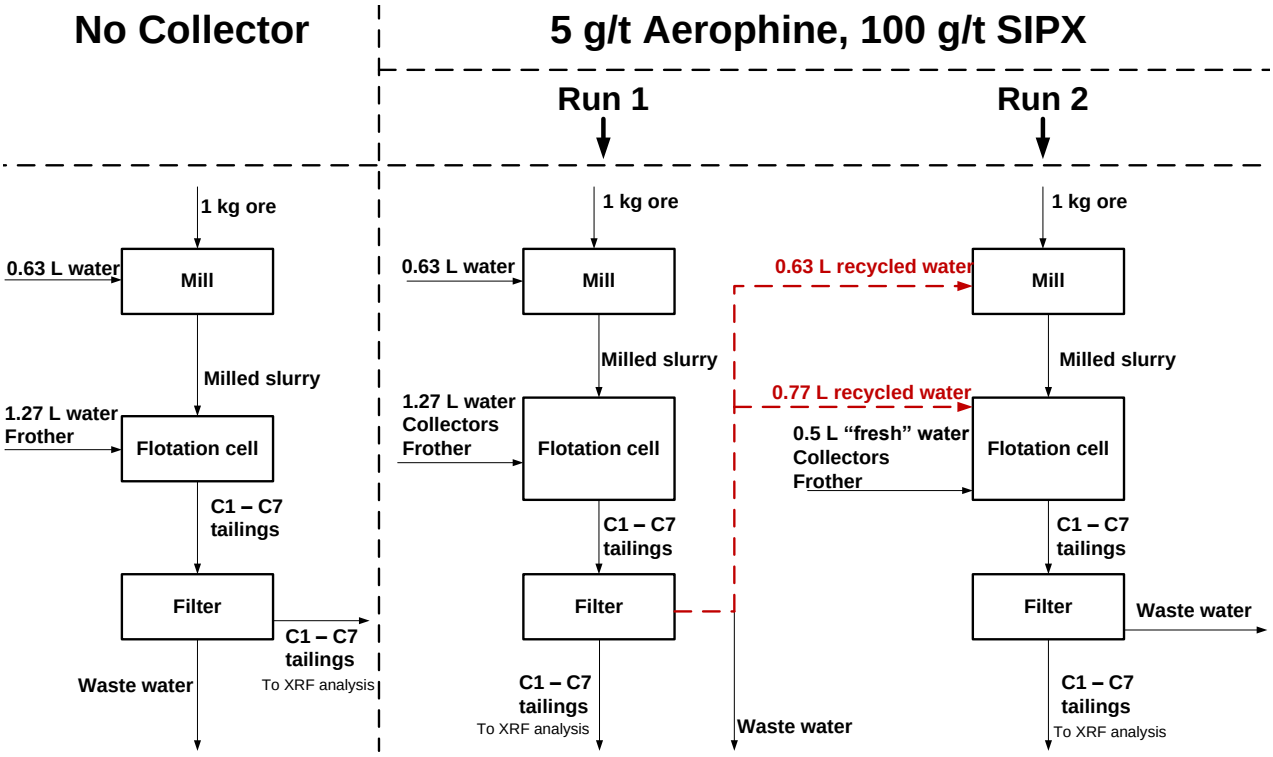

Figure 4.10: the procedure followed in simulating the recycle runs in the flotation of Ore B

4.5.3. Adsorption Kinetic Tests

Before the concentrates and tailings were filtered, supernatant samples were taken from them in order to determine the amount of collector still remaining in solution. This was done in the following way: SIPX and SIBX each have an absorbance peak at 301 nm; thus, UV/Vis spectrometry was used to determine the absorbance of each supernatant at 301 nm. From the absorbance, the xanthate concentration was then calculated using Beer-Lambert's Law (Equation 4.4), which expresses absorbance as directly proportional to concentration (Yates, 2012).

$$A = \varepsilon \ell c \qquad 4.4$$

In Equation 4.4, $\boldsymbol{A}$ is the dimensionless absorbance of a solution; $\boldsymbol{c}$ is the concentration of a compound in solution, measured in mol/L; $\boldsymbol{\ell}$ is the path length of the cuvette in which the sample is contained when analysed by UV/Vis spectrometry, measured in centimetres, and $\boldsymbol{\varepsilon}$ is the molar absorptivity, measured in L/mol.cm. The path length of a cuvette is constant, and is often 1 cm. Additionally, molar absorptivity, or the amount of light absorbed per unit concentration, is constant for a given substance; hence, the $\boldsymbol{\varepsilon\ell}$ term becomes constant. Since the Beer-Lambert Law is in the linear form $\boldsymbol{y = mx}$, $\boldsymbol{\varepsilon\ell}$ can be expressed as the slope of a graph plotting concentration against absorbance (Yates, 2012).

In order to obtain the graph plotting concentration against absorbance, standard solutions of known SIBX concentrations were prepared, and their absorbances determined. Figures 4.11 and 4.12 depict the respective absorbance curves of different concentrations of SIBX in distilled water and 1SPW.

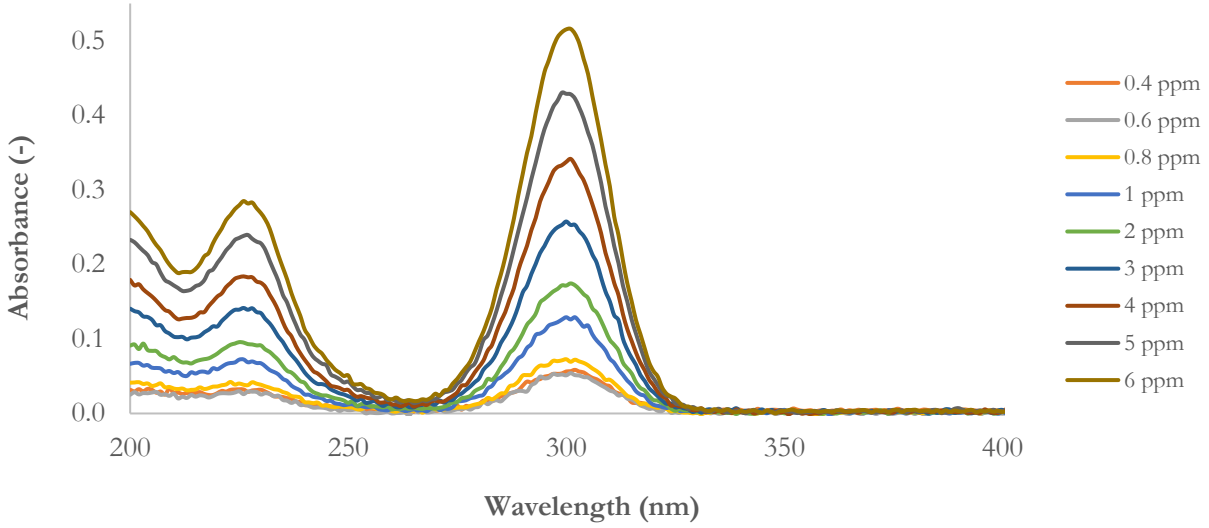

Figure 4.11: Absorbance of SIBX at different concentrations in distilled water

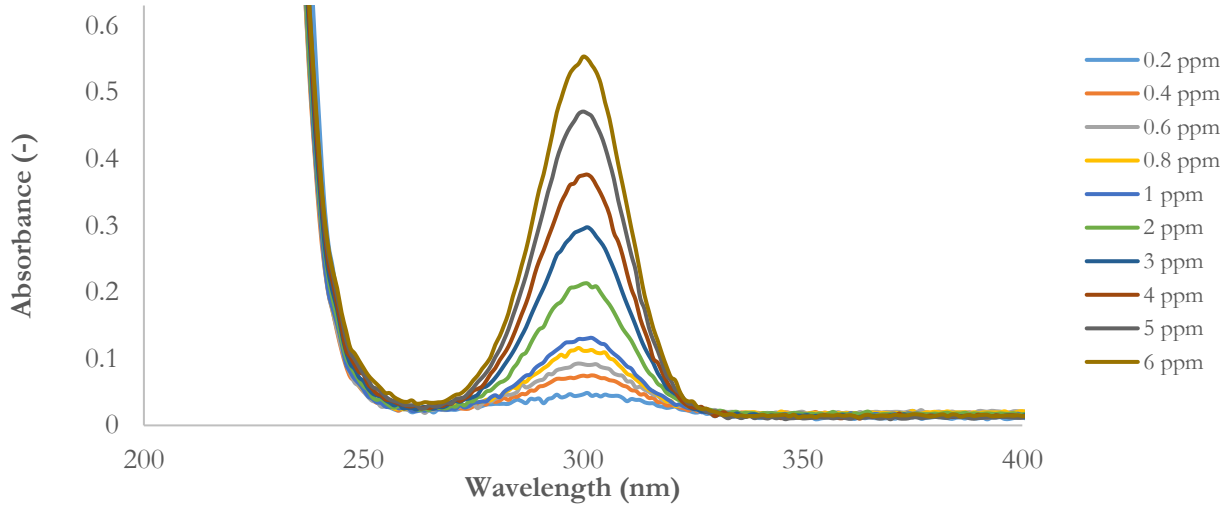

Figure 4.12: Absorbance of SIBX at different concentrations in 1SPW

Similar absorbance curves were obtained for 3SPW and 5SPW. From this, and for each water type, a calibration curve was drawn for a concentration range of the same order of magnitude. For example, two calibration curves were plotted for 1SPW, with one curve lying in the concentration range 0.2 to 1 ppm SIBX in solution, and the second curve in the concentration range 2 to 6 ppm SIBX in solution. Figure 4.13 shows calibration curves for distilled water and 1SPW, for 2 to 6 ppm SIBX solution.

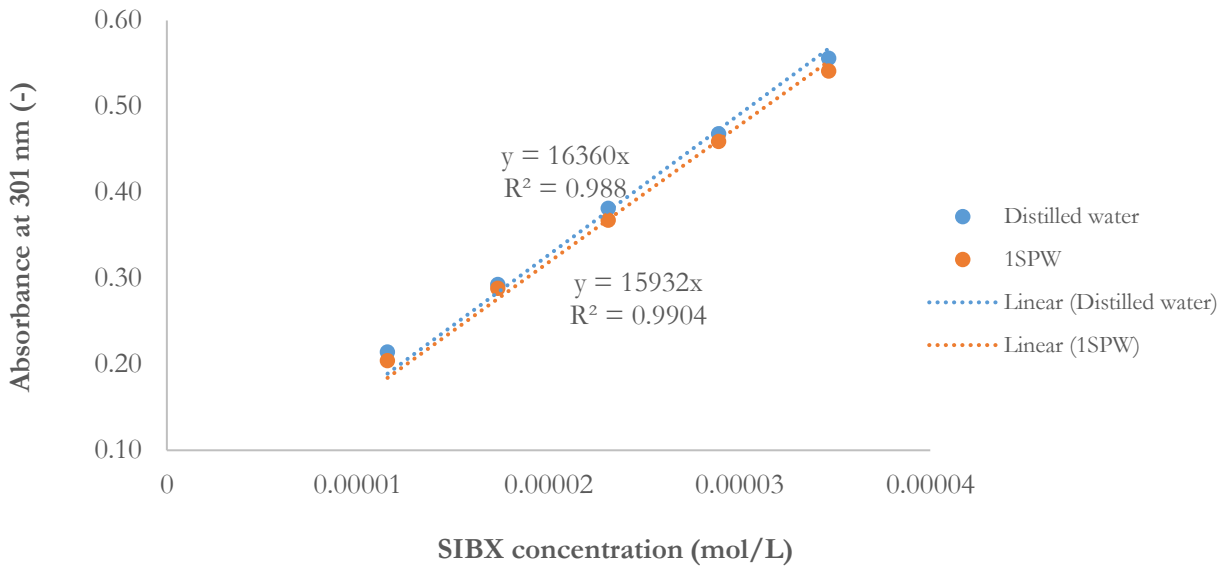

Figure 4.13: Calibration curves for distilled water and 1SPW for the SIBX concentration range of 2 to 6 ppm

The supernatants from the concentrates and tailings had unknown concentrations of xanthate and so, by rearranging Equation 4.4 and using the respective slope values obtained from the calibration curves, their concentrations were determined using Equation 4.5 and Equation 4.6.

$$c = \frac{A}{\varepsilon \ell} \qquad 4.5$$

$$c = \frac{A}{\text{slope of calibration curve}} \qquad 4.6$$

Given that the initial collector concentration was known, and the amount of non-adsorbed collector could be determined, the amount of collector adsorbed onto the mineral particles was indirectly determined as the balance between these two values (Hadler et al., 2005; Raju and Forsling, 1991).

4.5.4. Feed, Tails and Concentrate Analysis

The respective copper and nickel amounts contained in each concentrate, feed and tailings sample were used as the proxy contents of chalcopyrite and pentlandite. This was done with the assumption that the stoichiometry of chalcopyrite is $CuFeS_2$, and that of pentlandite is $(FeNi)_9S_8$ (Corin et al., 2011). For each concentrate, the mass of gangue minerals was determined as the difference between the total mass of the concentrate and the total mass of the valuable minerals.

5. RESULTS

This chapter presents the results of the conducted experiments, the key objectives of which, were to determine the effect of water quality on xanthate adsorption, and further, to investigate the effect of the adsorption on the flotation responses of two sulphide ores. The flotation responses were evaluated with the following as key performance parameters: the recovery and grade of copper (for Ore A), the recovery and grade of copper and nickel (for Ore B), the recovery of water, and the recovery of total solids and unwanted gangue materials. The results will be presented in two sections. Section 5.2. will discuss results pertaining to Ore A, while Section 5.3. will present those pertaining to Ore B.

5.1. Reproducibility of the Tests

All experiments were performed in duplicate. The precision of the tests was evaluated by a standard error analysis, which is represented by error bars on all presented graphs.

5.2. The Effects of Water Quality on the Flotation Response of Ore A

The effect of water quality on the flotation of Ore A was tested at 0, 50 and 100 g/t SIBX. For 50 and 100 g/t SIBX tests, two batches of experiments were conducted in order to simulate water recycling. In the first batch, Run 1 was dossed with either 50 or 100 g/t SIBX; for each initial dosage, the first recycle run (Run 2) was carried out without additional collector being dosed, and the second recycle run (Run 3) was also carried out without additional collector. In the second batch, say, for 50 g/t collector dosage, Run 1 was dosed with 50 g/t SIBX; Run 2 was dosed with 50 g/t SIBX, and Run 3 was also dosed with 50 g/t SIBX. These conditions are summarised in Figure 5.1.

	Baseline	No SIBX dosed in recycle runs		Additional SIBX dosed in recycle runs	
Run 1	0 g/t	50 g/t	100 g/t	50 g/t	100 g/t
Run 2	-	Additional 0 g/t	Additional 0 g/t	Additional 50 g/t	Additional 100 g/t
Run 3	-	Additional 0 g/t	Additional 0 g/t	Additional 50 g/t	Additional 100 g/t

Figure 5.1: The collector dosages tested at different ionic strengths for Ore A

5.2.1. The Flotation Response of Ore A in Collectorless Water

For flotation without added collector, Figure 5.2 and Figure 5.3 illustrate the respective effects of water quality on the recovery of water and the recovery of solids to the concentrate.

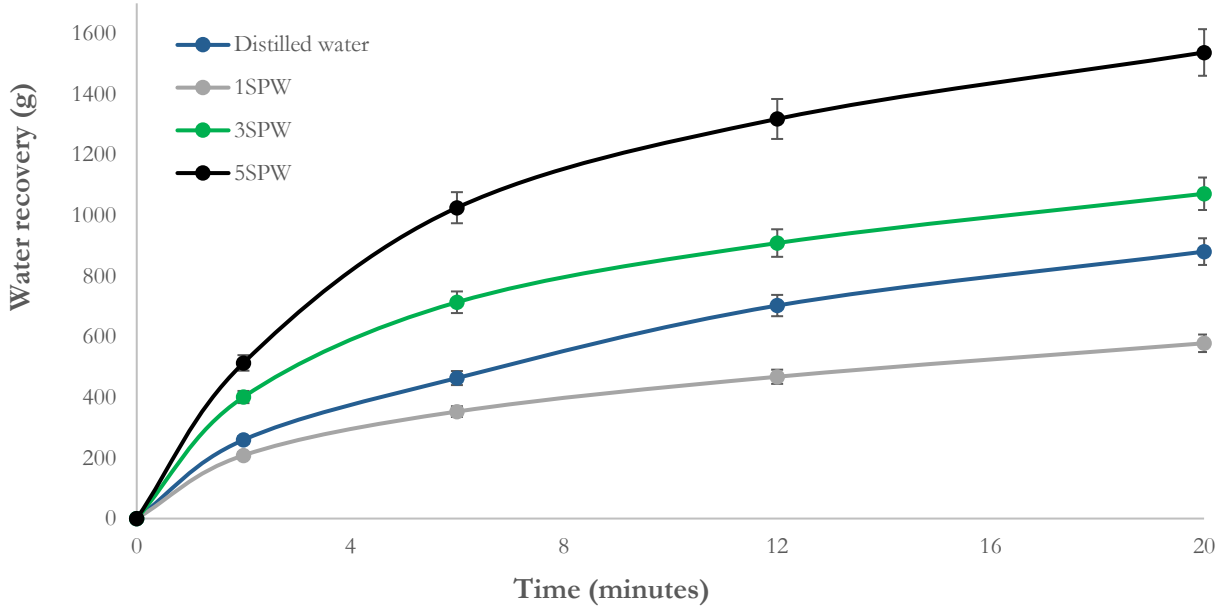

Figure 5.2: Water recovery vs. time for Ore A; no collector and all tested waters

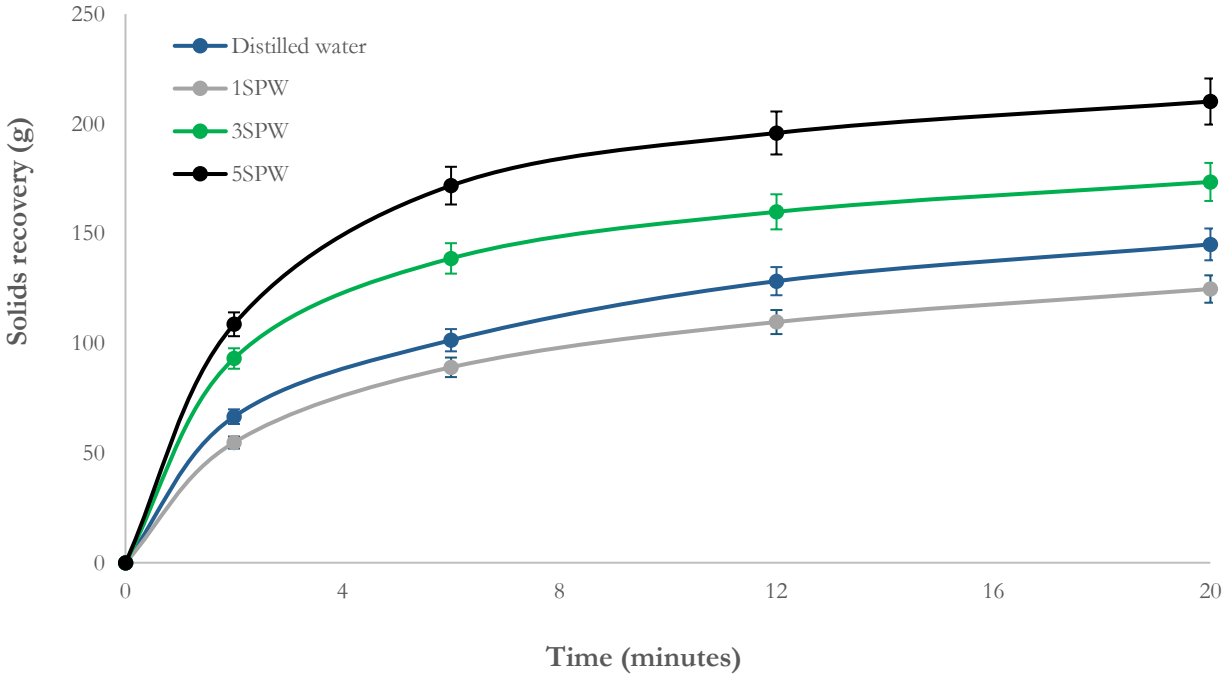

Figure 5.3: Solids recovery vs. time for Ore A; no collector and all tested waters

Figure 5.4 depicts the effect of water quality on the total recovery of water, solids and gangue materials; and Figure 5.5 depicts the recovery of solids to the concentrate as a function of water recovery.

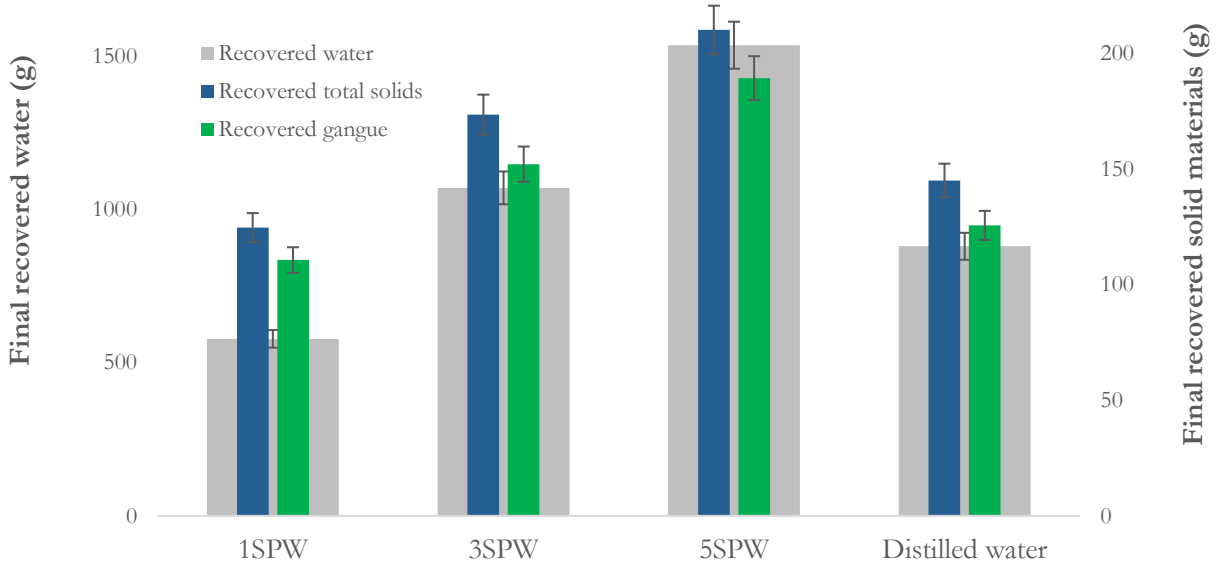

Figure 5.4: Total solids and gangue material vs. water recovered for Ore A; no collector and all tested waters

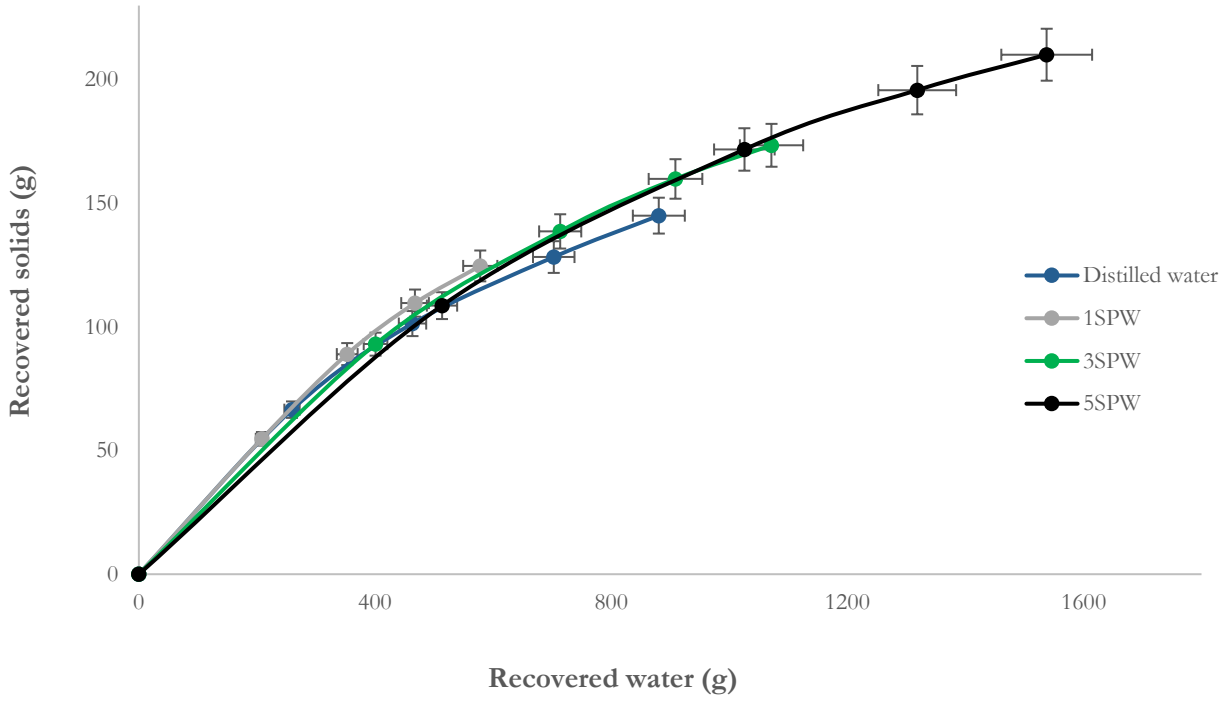

Figure 5.5: Solids vs. water recovery for Ore A; no collector and all tested waters

The observed trend is that as water recovery to the concentrate increased, so did the mass of solids (and thus gangue minerals). Further, as the ionic strength increased, so did the recovery of total

amounts of water and solids; but the recoveries observed in distilled water were higher than those observed in 1SPW, and lower than those observed in 3SPW.

Figure 5.6 illustrates the recovery of copper to the concentrate as a function of flotation time, for no added collector and in all tested waters. Figure 5.7 illustrates the copper grade vs. recovery curve; and Figure 5.8 illustrates the final grade and recovery. From these, it can be seen that the highest final copper recovery was achieved in 5SPW flotation (67.5%), while the highest final grade was achieved in distilled water (4.61%). At the same time, owing to the high solids (and gangue mineral recovery), the lowest final grade was achieved in 5SPW flotation.

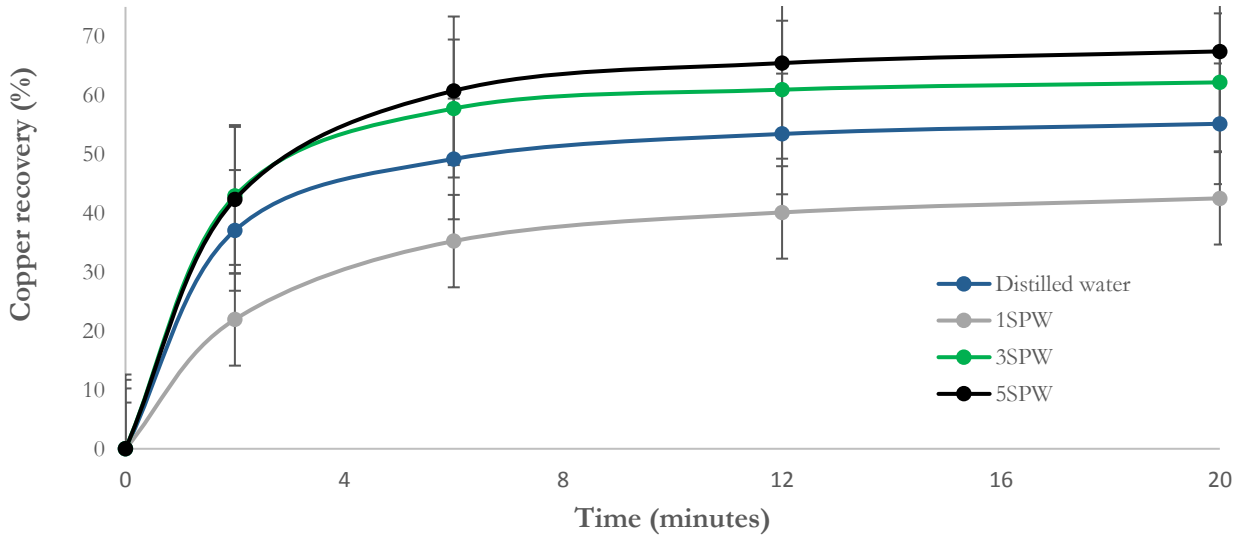

Figure 5.6: Copper recovery vs. time for Ore A; no collector and all tested waters

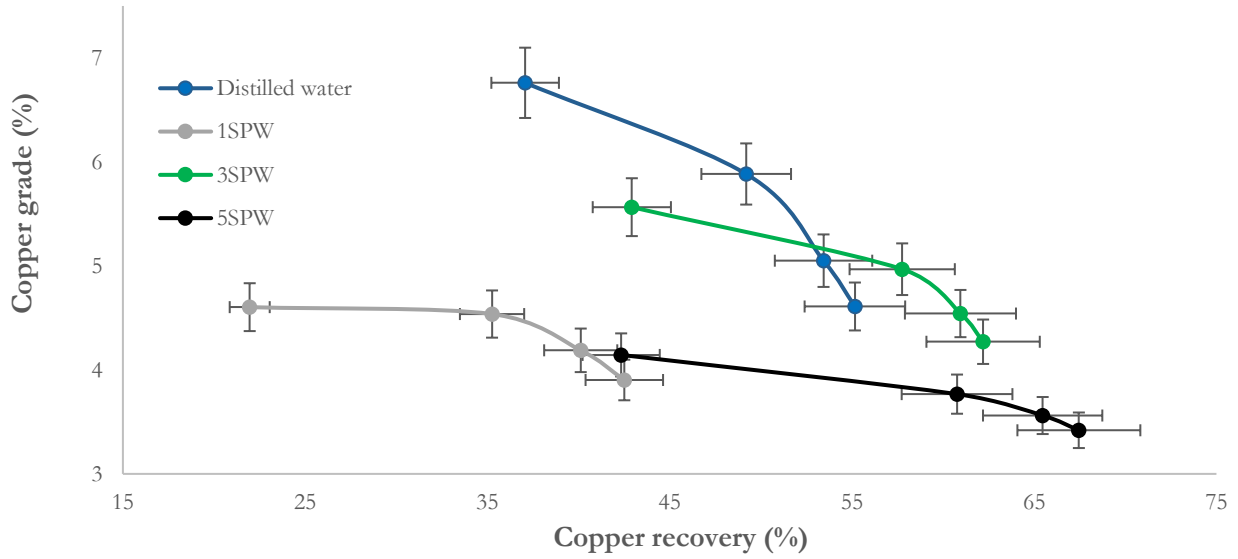

Figure 5.7: Copper grade vs. recovery for Ore A; no collector and all tested waters

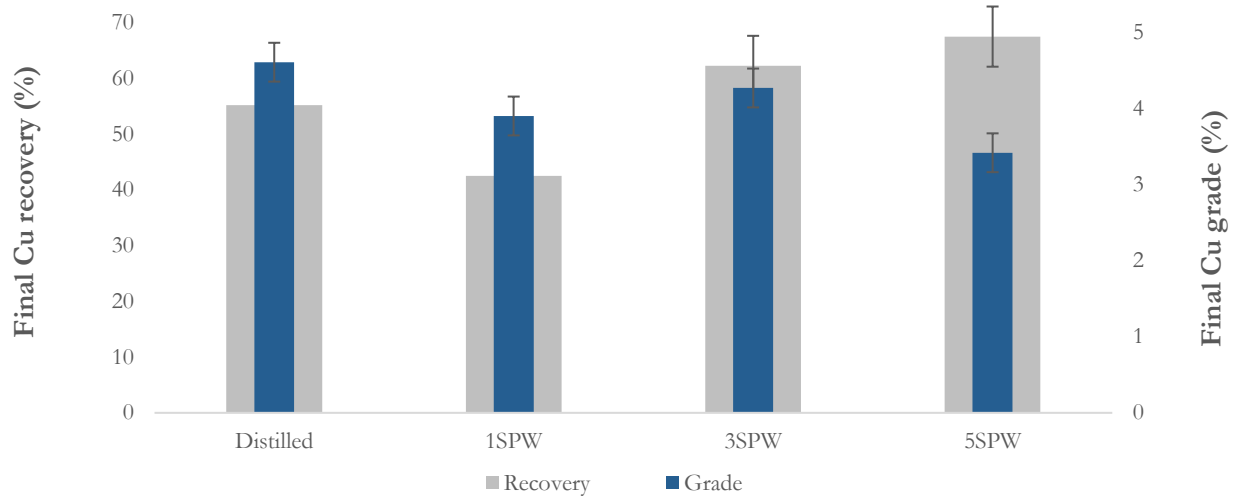

Figure 5.8: Final copper grade vs. recovery for Ore A; no collector and all tested waters

Thus, for flotation of Ore A with no collector, as the ionic strength of the water increased, so did the recovery of solids and water to the concentrate. The recovery of copper was also increased, and the distilled water tests yielded the highest grade while 5SPW yielded the lowest.

5.2.2. The Flotation Response of Ore A in Non-Recycled Water (Run 1)

For a dosage of 50 and 100 g/t SIBX, Figure 5.9 illustrates the effect of water quality on the recovery of water to the concentrate, and Figure 5.10 illustrates effect of water quality on the recovery of solids.

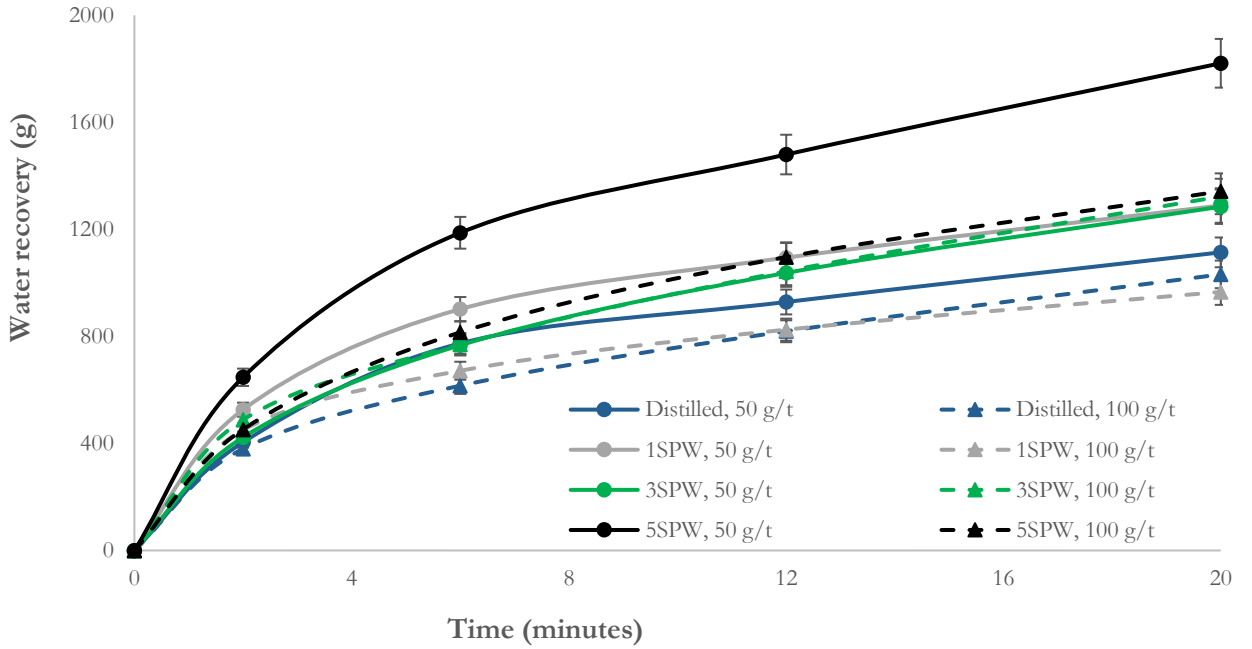

Figure 5.9: Water recovery vs. time for Ore A; 50 and 100 g/t SIBX, in all tested waters

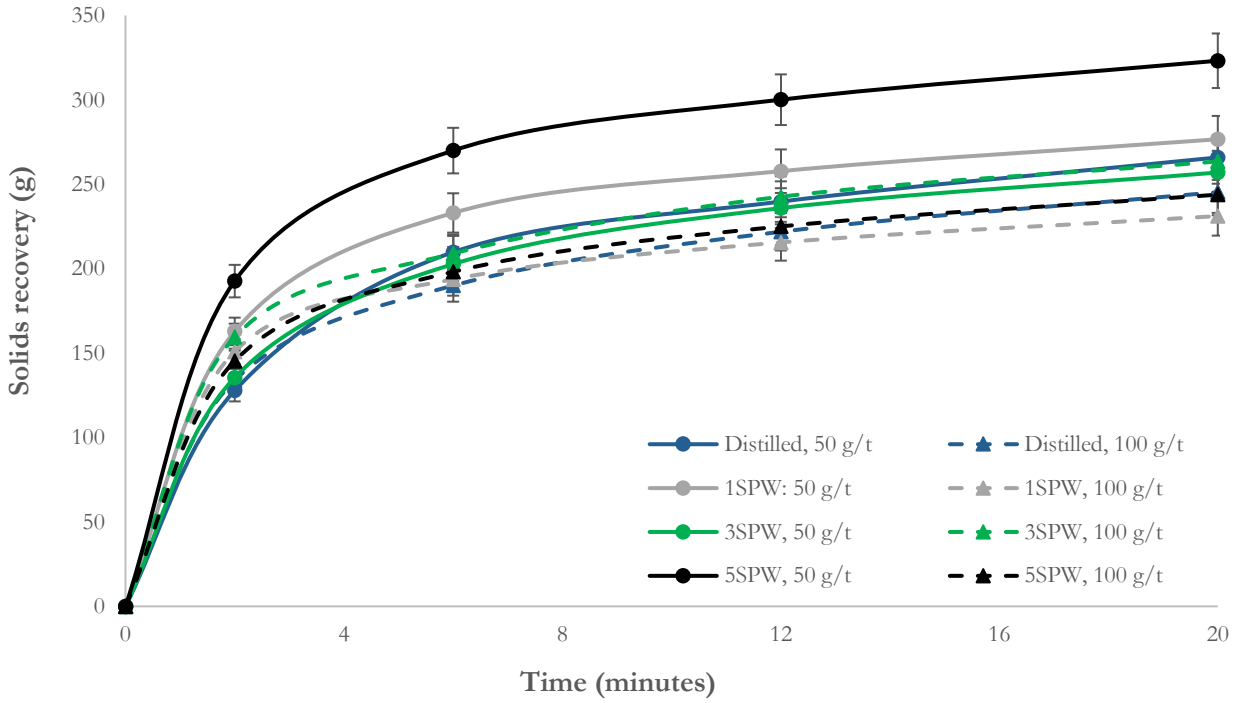

Figure 5.10: Solids recovery vs. time for Ore A; 50 and 100 g/t SIBX, in all tested waters

Figure 5.11 depicts the effect of water quality on the total recovery of water, solids and gangue materials; and Figure 5.12 depicts solids recovery as a function of water recovery.

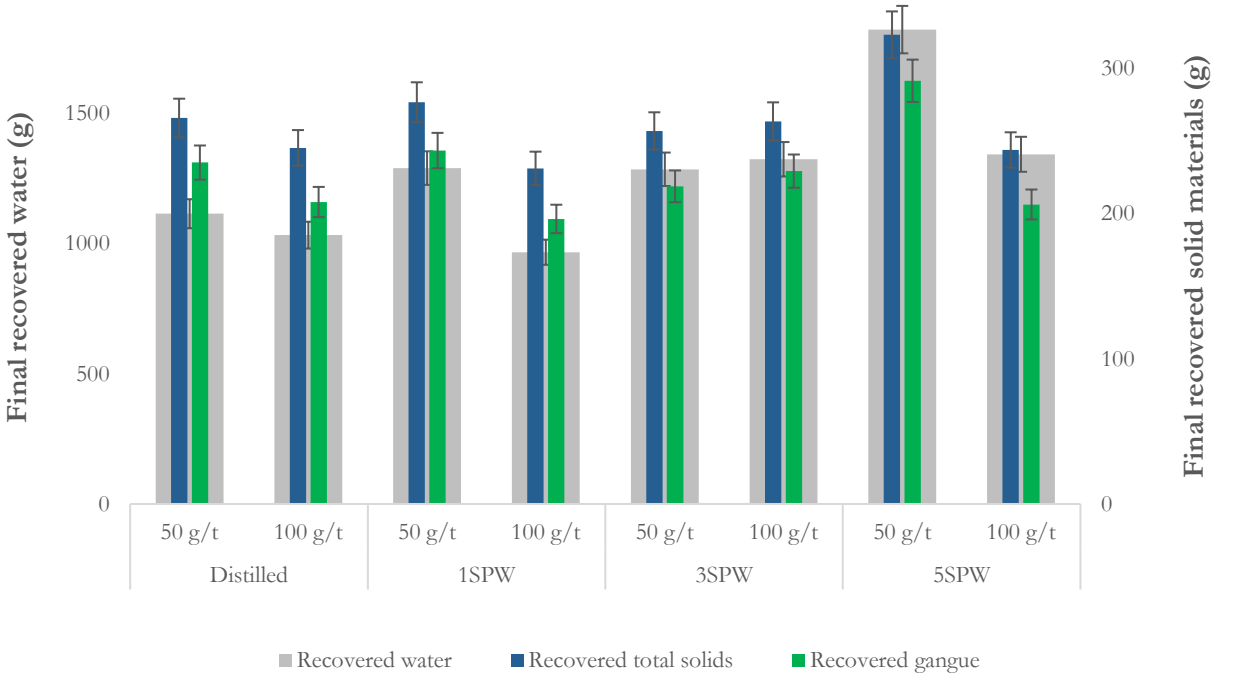

Figure 5.11: Total solids and gangue material vs. water recovered for Ore A; 50 and 100 g/t SIBX, in all tested waters

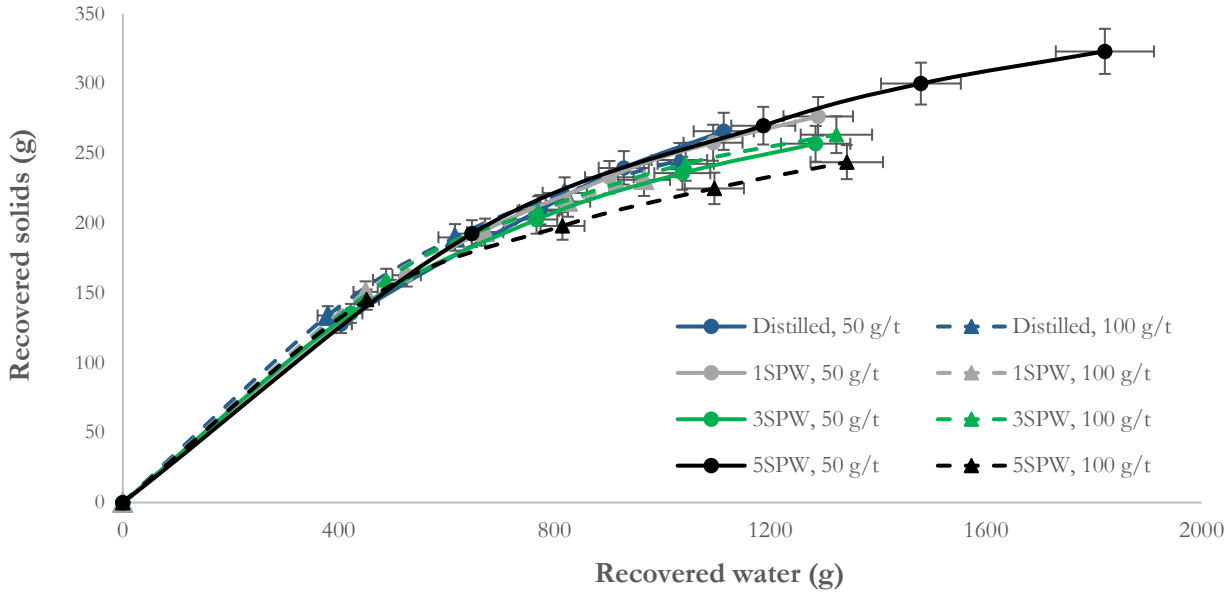

Figure 5.12: Solids vs. water recovery for Ore A; 50 and 100 g/t SIBX, in all tested waters

As observed with the collectorless trends, increased water recoveries were complemented by increased solid material recoveries. However, when compared with the collectorless runs, the runs floated with SIBX yielded more total water, solids and gangue materials. Moreover, for all tested waters (with the exception of 3SPW) less total water and solids were recovered for 100 g/t SIBX than for 50 g/t. Further, flotation in 50 g/t SIBX in 5SPW resulted in the highest water and solids recovery, while flotation in 100 g/t SIBX in 1SPW resulted in the lowest. The copper recoveries and grades were improved when compared to the collectorless tests, as demonstrated by Figure 5.13 and Figure 5.14.

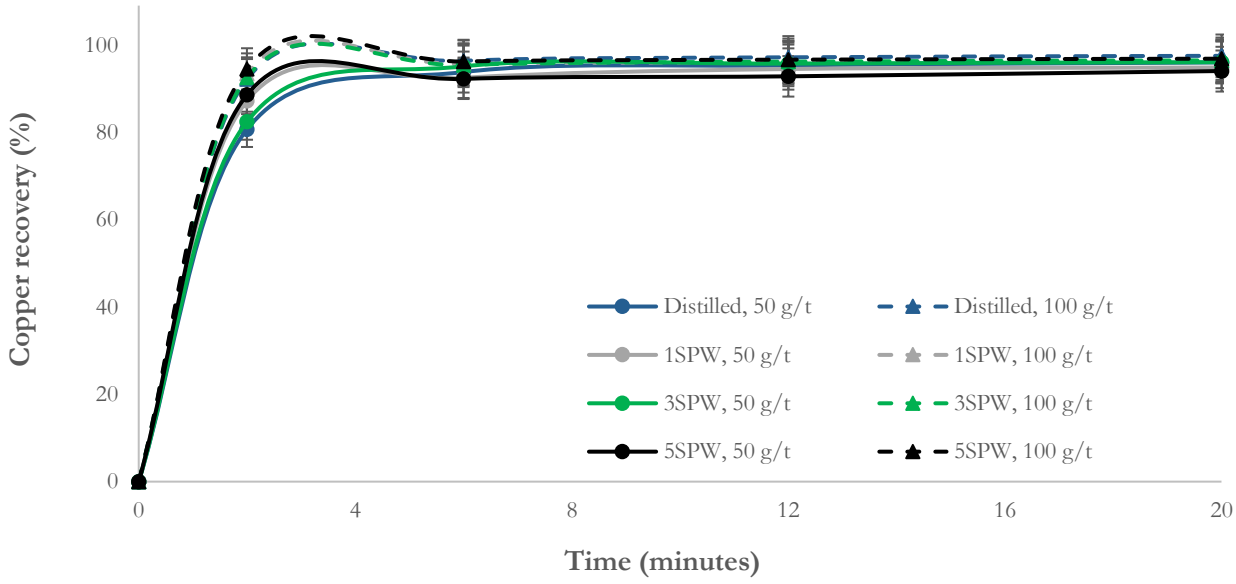

Figure 5.13: Copper recovery vs. time for Ore A; 50 and 100 g/t SIBX, in all tested waters

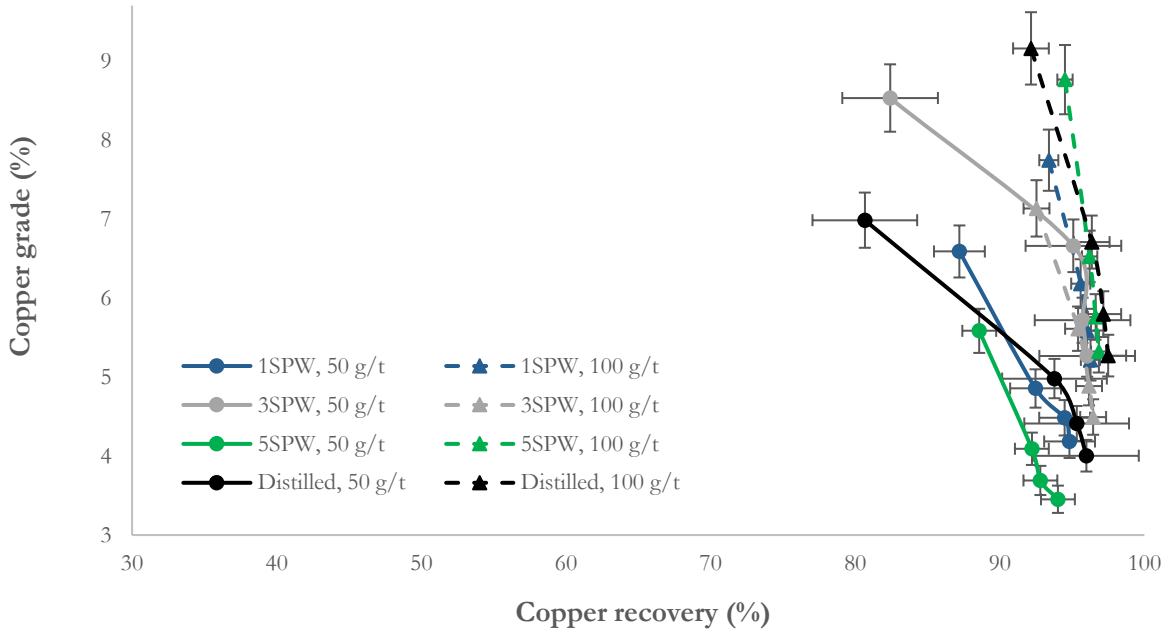

Figure 5.14: Copper grade vs. recovery for Ore A; 50 and 100 g/t SIBX, in all tested waters

The copper recoveries were high such that, for the 50 g/t dosage, the initial recoveries for all tested waters were above 80%, and the final recoveries were above 94%. For the 100 g/t dosage, each tested water type had an initial recovery above 92%, and a final recovery above 94%. This suggests that for both collector dosages, final copper recovery was not affected by water quality. The result is that the recovery vs. time and the grade vs. recovery curves cluster to one extreme end. As such, the final values, especially the obtained grades, are better depicted in bar-form, presented in Figure 5.15.

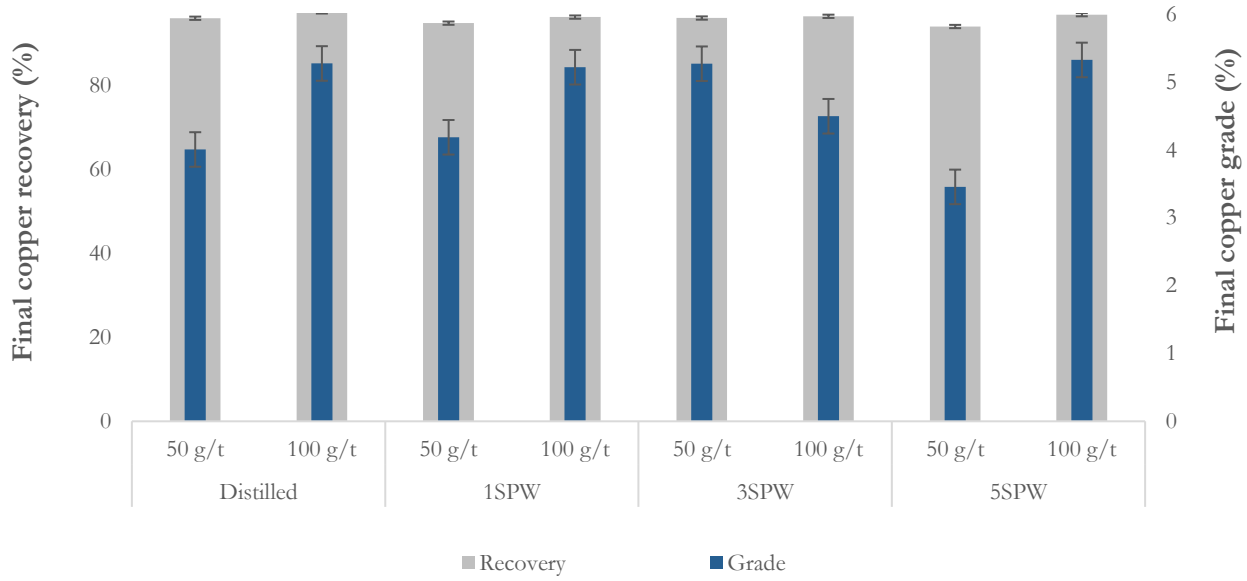

Figure 5.15: Final copper grade vs. recovery for Ore A; 50 and 100 g/t SIBX, in all tested waters

For all tested waters (with the exception of 3SPW) better grades were observed in flotation under a dosage of 100 g/t SIBX than under a dosage of 50 g/t.

Figure 5.16 and Figure 5.17 show the SIBX concentration remaining in solution at the time intervals at which concentrates were collected, as well as the SIBX remaining in tailings; this, for an initial dosage of 50 and 100 g/t SIBX, respectively.

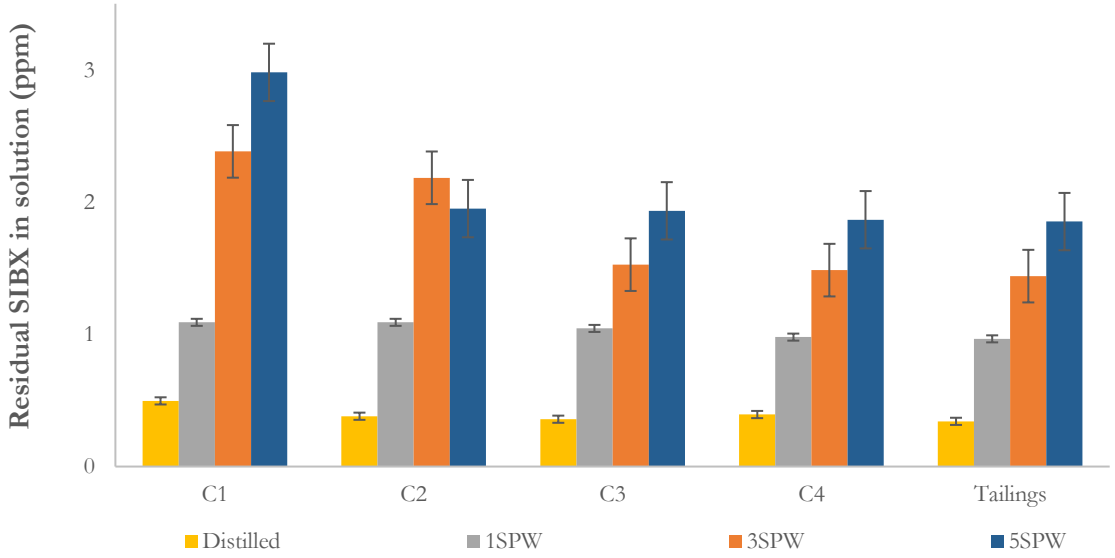

Figure 5.16: The collector concentration remaining in C1 to C4, and in the tailings, for an initial dosage of 50 g/t SIBX in Run 1

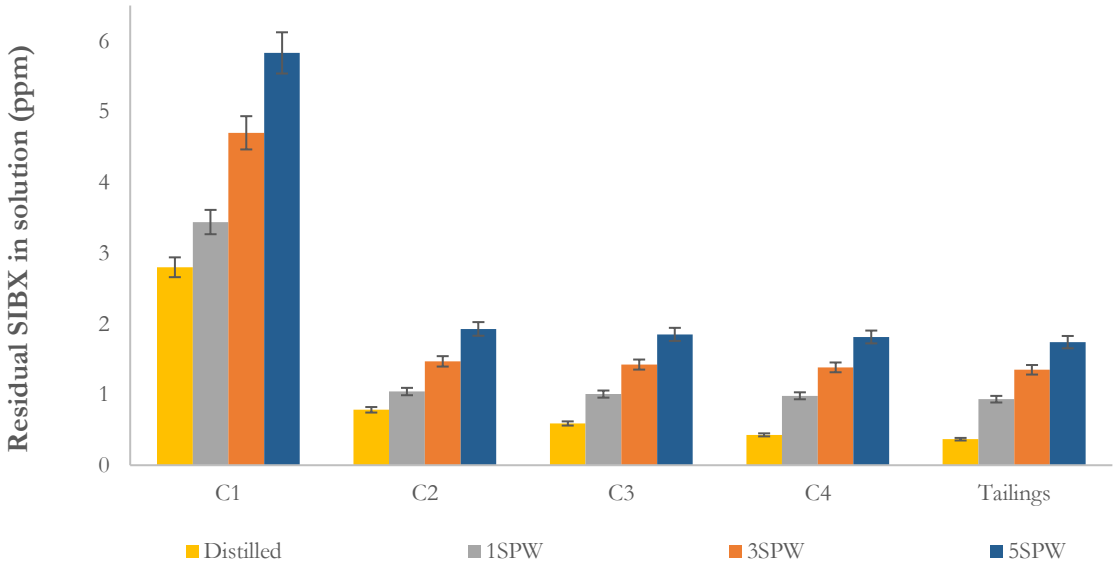

Figure 5.17: The collector concentration remaining in C1 to C4, and in the tailings, for an initial dosage of 100 g/t SIBX in Run 1

For Figure 5.16, the 50 g/t SIBX dosage is equivalent to an initial concentration of 17.2 ppm SIBX in solution, and for Figure 5.17, the 100 g/t dosage is equivalent to 34.5 ppm. For both dosages, and for all tested waters, at the end of flotation the greater fraction of the dosed SIBX had been adsorbed. However, it can still be noted that the least amount of SIBX was adsorbed at 100 g/t dosage in 5SPW (where 88.9% of the initial xanthate was adsorbed and 1.74 ppm remained in solution), and the highest amount was adsorbed in 100 g/t dosage in distilled water.

It can further be noted that near the beginning of flotation, when C1 was collected, the SIBX in solution was already much lower than the initial concentration, but still higher than at the subsequent time intervals. At the time of collection for C2 to C4, the respective residual concentrations have little difference between them. This suggests that the greater amount of the collector was adsorbed in the beginning stages of flotation, after which little collector remained, and little adsorption occurred.

Hence, the overall adsorption trend can be summarised by Figure 5.18 and Figure 5.19, where the former summarises the residual SIBX concentration in C1 and the tailings of each water, and the latter summarises the fraction of the initial concentration adsorbed in each tested case. Further, from Figure 5.19, it is apparent that across the different tested waters, and for each collector dosage, the adsorbed amount and adsorbed fraction can each be approximated by a linear relationship. The respective R-squared values are close to 1 (at 0.989 for the 50 g/t dosage and 0.955 for the 100 g/t dosage). This indicates sufficient fits for the respective linear approximations.

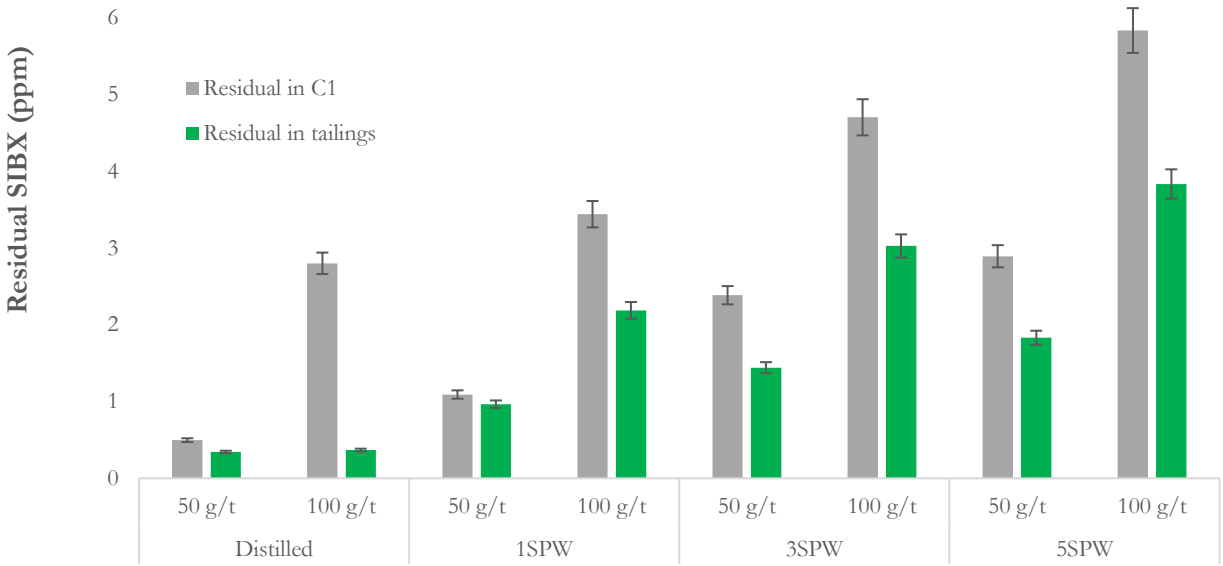

Figure 5.18: A summary of the amount of adsorbed SIBX in Run 1

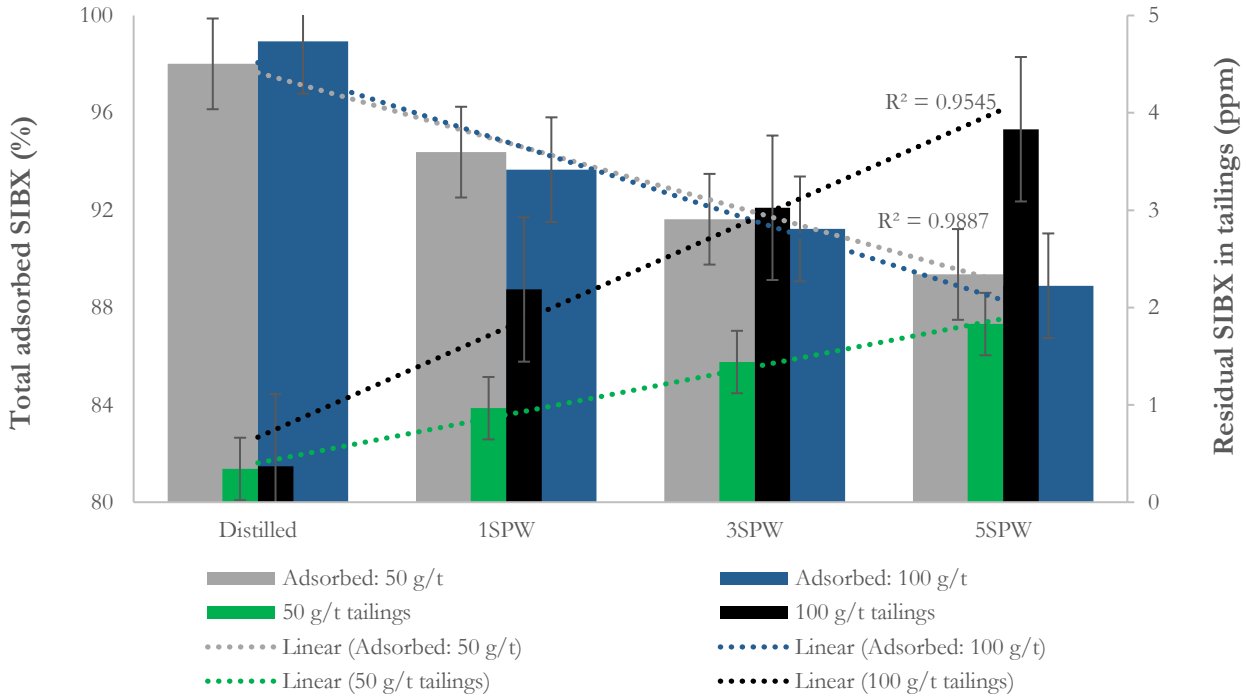

Figure 5.19: A summary of the amount of adsorbed SIBX in mg/litres (=ppm), plotted against the fraction of the initial dosage that was adsorbed in Run 1

Figure 5.20 plots the amount of xanthate that was adsorbed against the final water recovery in all tested water qualities, and Figure 5.21 plots the total adsorbed SIBX against the total amount of recovered solids.

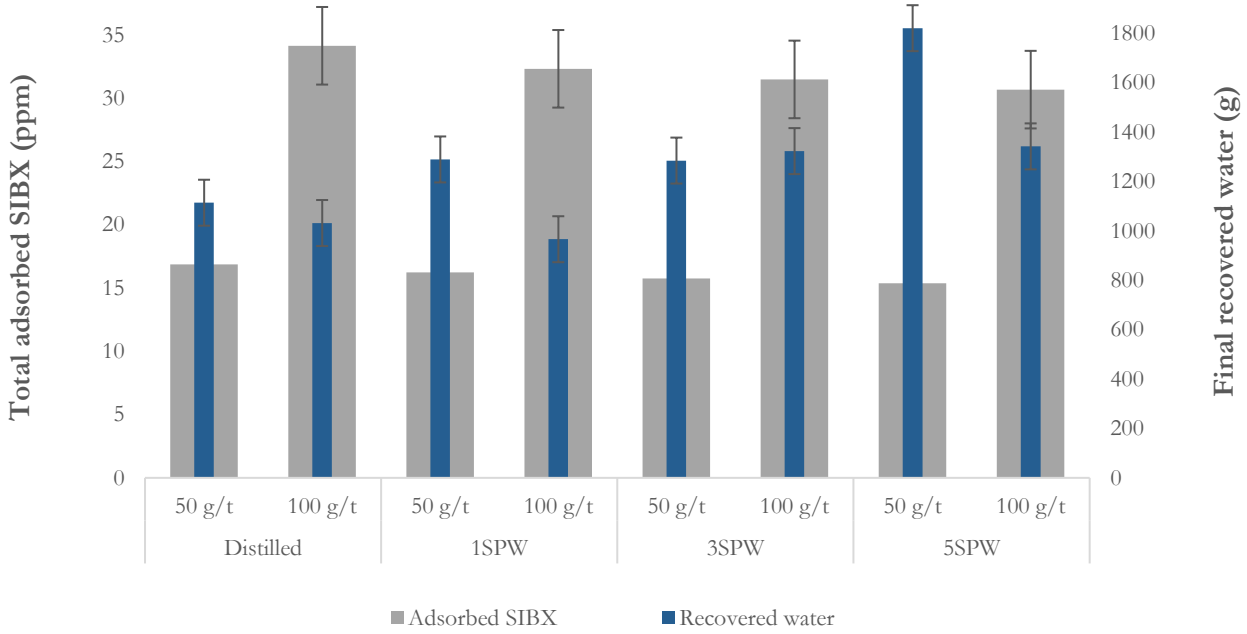

Figure 5.20: The total adsorbed SIBX plotted against final water recovery for all tested waters in Run 1

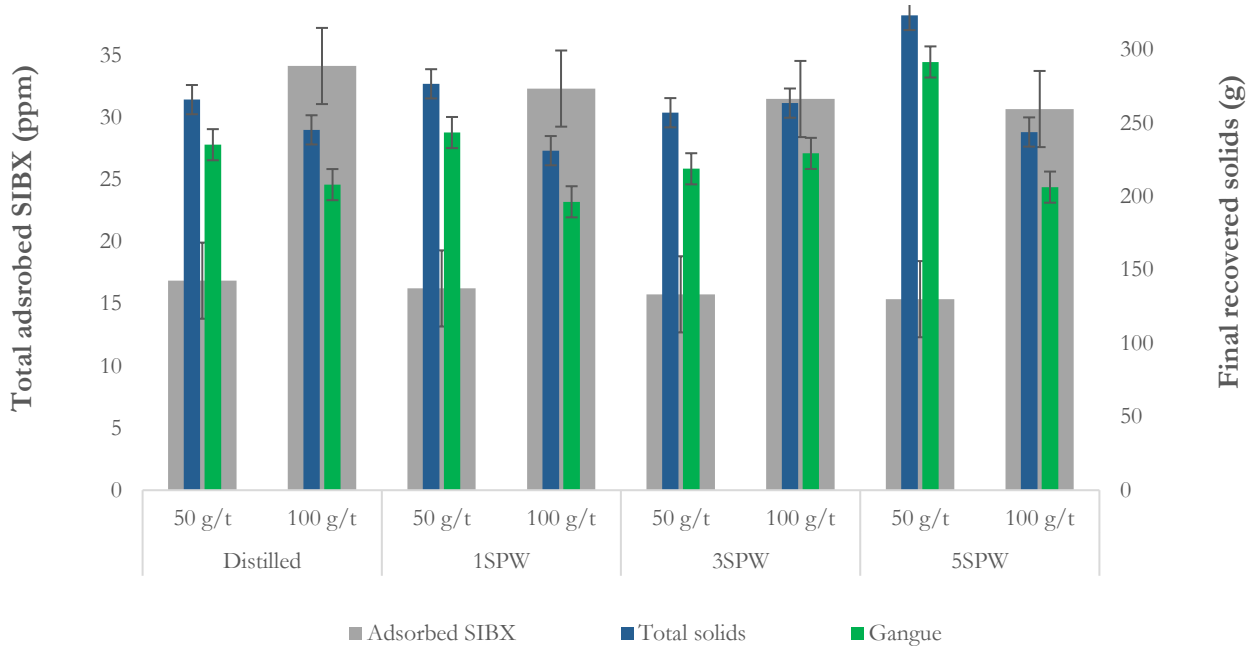

Figure 5.21: The total adsorbed SIBX plotted against final solids recovery for all tested waters in Run 1

For distilled water, 1SPW and 5SPW, more water was recovered when less SIBX was adsorbed, while for 3SPW, more water was recovered when more SIBX was adsorbed. For distilled water, 1SPW and 5SPW, more solids were recovered when less SIBX was adsorbed, while for 3SPW, more solids were recovered when more SIBX was adsorbed.

Figure 5.22 shows the final copper recoveries against the total amount of adsorbed xanthate. The figure suggests that the degree of collector adsorption did not affect final copper recovery.

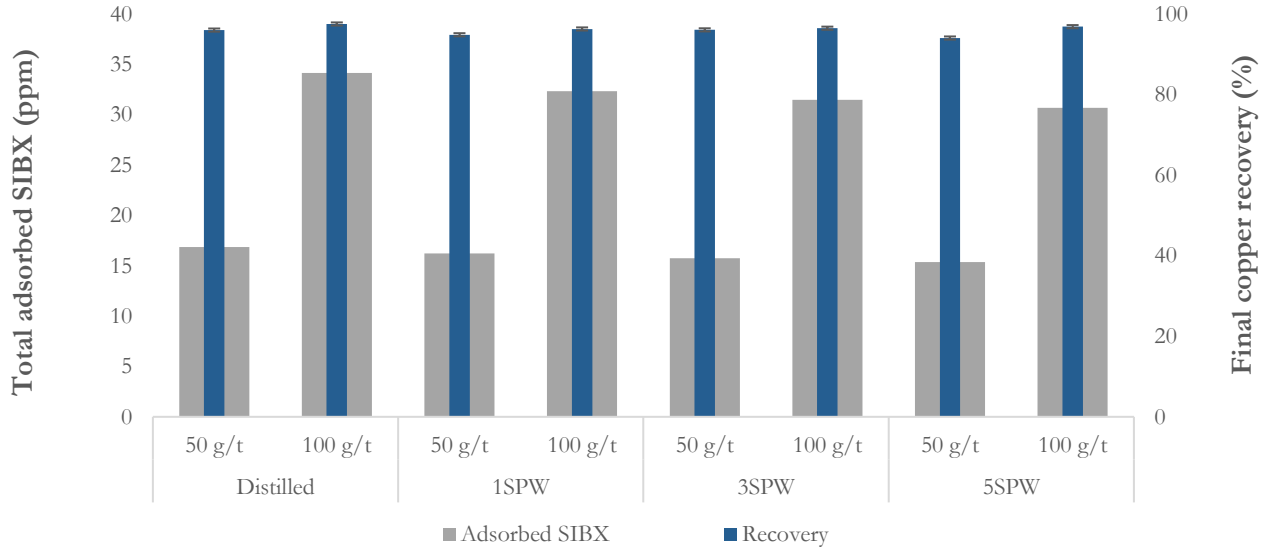

Figure 5.22: The total adsorbed SIBX plotted against final copper recovery for all tested waters in Run 1

Figure 5.23 shows the final copper grades against the total amount of adsorbed SIBX. From the figure, as more xanthate was dosed, and as more xanthate was adsorbed, the concentrate grades improved; this, in all cases except for 3SPW. This suggests that for distilled water,1SPW and 5SPW, higher xanthate adsorption resulted in improved concentrate grades.

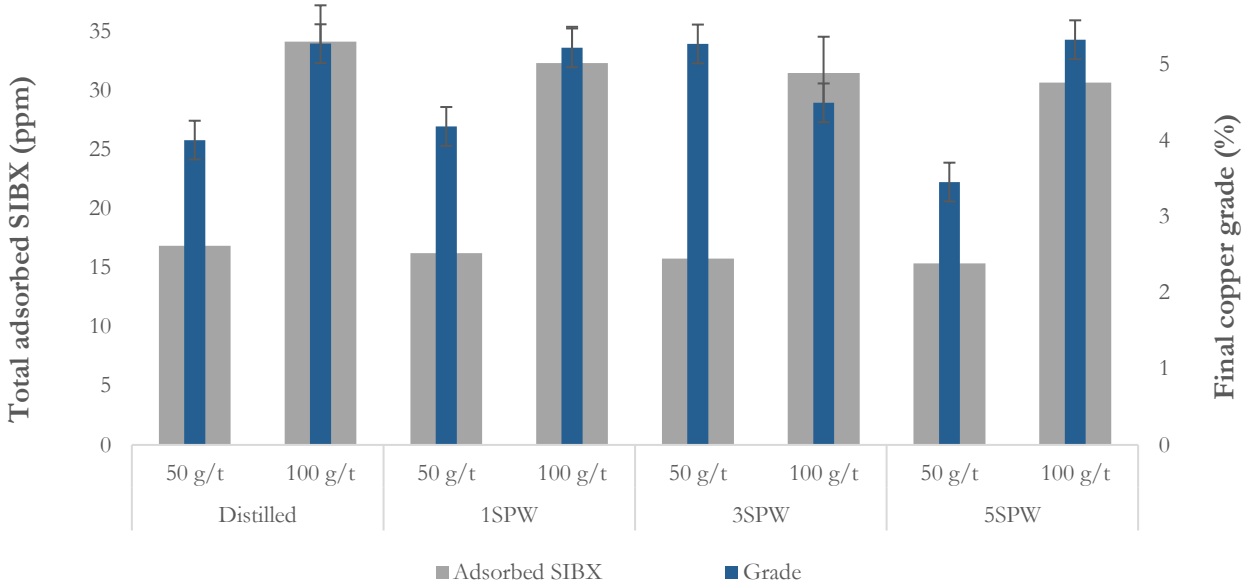

Figure 5.23: The total adsorbed SIBX plotted against final copper grade for all tested waters in Run 1

5.2.3. The Flotation Response of Ore A in Once-Recycled Water (Run 2)

For Run 2, for the first batch of tests, i.e. the recycle runs without additional collector, the key for the results (in the figures and tables presented overleaf) is as follows: *50,0* for tests in which Run 1 was dosed with 50 g/t SIBX and Run 2 was not, and *100,0* for tests in which Run 1 was dosed with 100 g/t and Run 2 was not. For the second batch of tests, i.e. recycle runs with additional collector, the key for the results is as follows: *50,50* for tests where Run 1 and Run 2 were each dosed with 50 g/t SIBX, and *100,100* for tests where Run 1 and Run 2 were each dosed with 100 g/t.

As a result of different concentrations of SIBX being adsorbed in Run 1, different concentrations of SIBX were therefore present at the start of the recycle runs. Since, for the synthetic plant waters, and as discussed in Section 5.2.2., the least amount of collector was adsorbed in 5SPW-Run 1, while the most was adsorbed in 1SPW-Run 1, it was thus the case that for Run 2, flotation in 5SPW-Run 2 commenced with the most amount of SIBX when compared to the other waters, and flotation in 1SPW-Run 2 commenced with the least amount. For the different waters, Table 5.1 shows the collector concentrations at the beginning of Run 2.

Table 5.1: SIBX concentrations in the feed for the recycle runs (Run 2)

Water	1SPW	3SPW	5SPW
SIBX in *50,0* (ppm)	0.599	0.883	1.14
SIBX in *50,50* (ppm)	17.8	18.1	18.4
SIBX in *100,0* (ppm)	0.579	0.838	1.09
SIBX in *100,100* (ppm)	35.1	35.3	35.6

Figure 5.24 illustrates the effect of water quality on the recovery of water to the concentrate, and Figure 5.25 illustrates effect of water quality on the recovery of solids. Figure 5.26 depicts the effect of water quality on the total recovery of water, solids and gangue materials; and Figure 5.27 depicts the recovery of solids to the concentrate as a function of water recovery.

The observed trend is that more total water was recovered in the recycle runs where no additional collector was dosed, but on the other hand, more total solids (and gangue materials) were recovered in the recycle runs where additional collector was dosed. This is true for all cases except *50,0* and *50,50* SIBX in 3SPW, where more solids as well as water masses were recovered for *50,50* SIBX. Moreover, when compared with Run 1, the recycle runs in which additional collector was dosed yielded lower solids and water recoveries; this, for all tested waters except 3SPW, where the opposite was true.

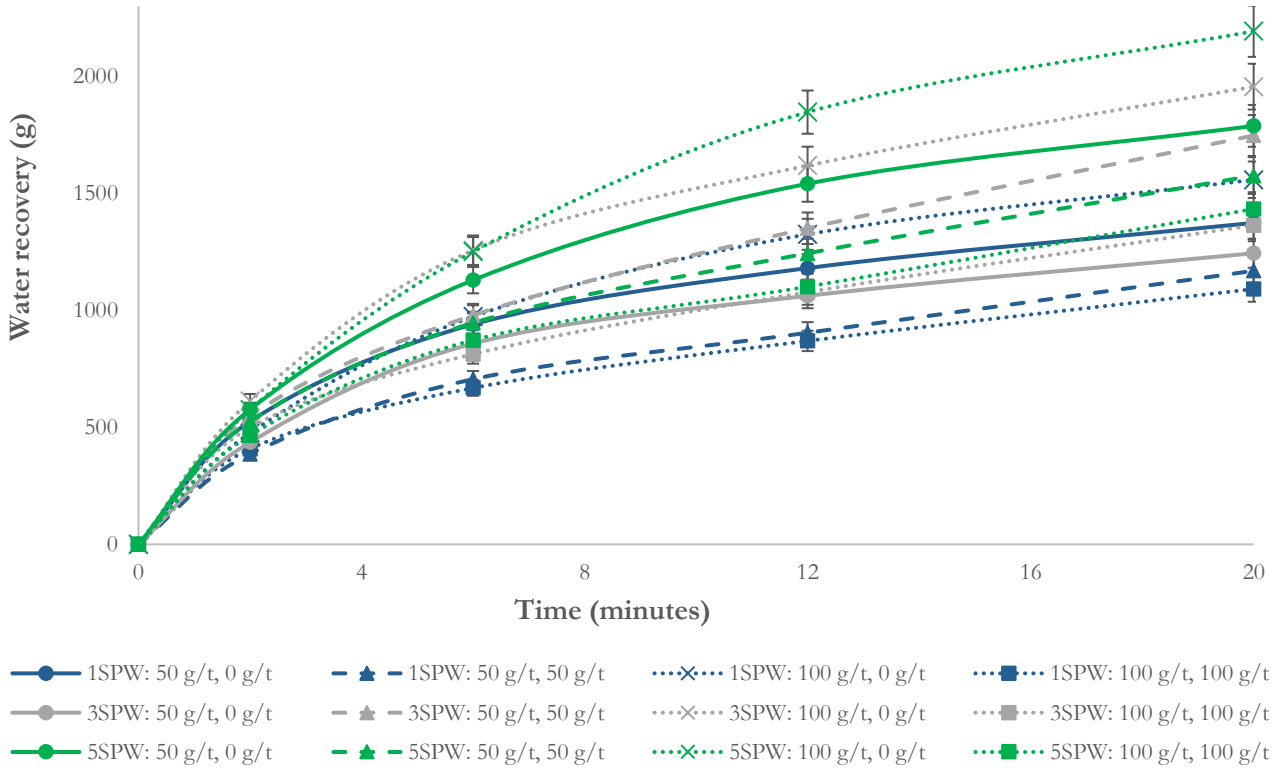

Figure 5.24: Water recovery vs. time for Ore A; 50 and 100 g/t SIBX, in all once-recycled waters

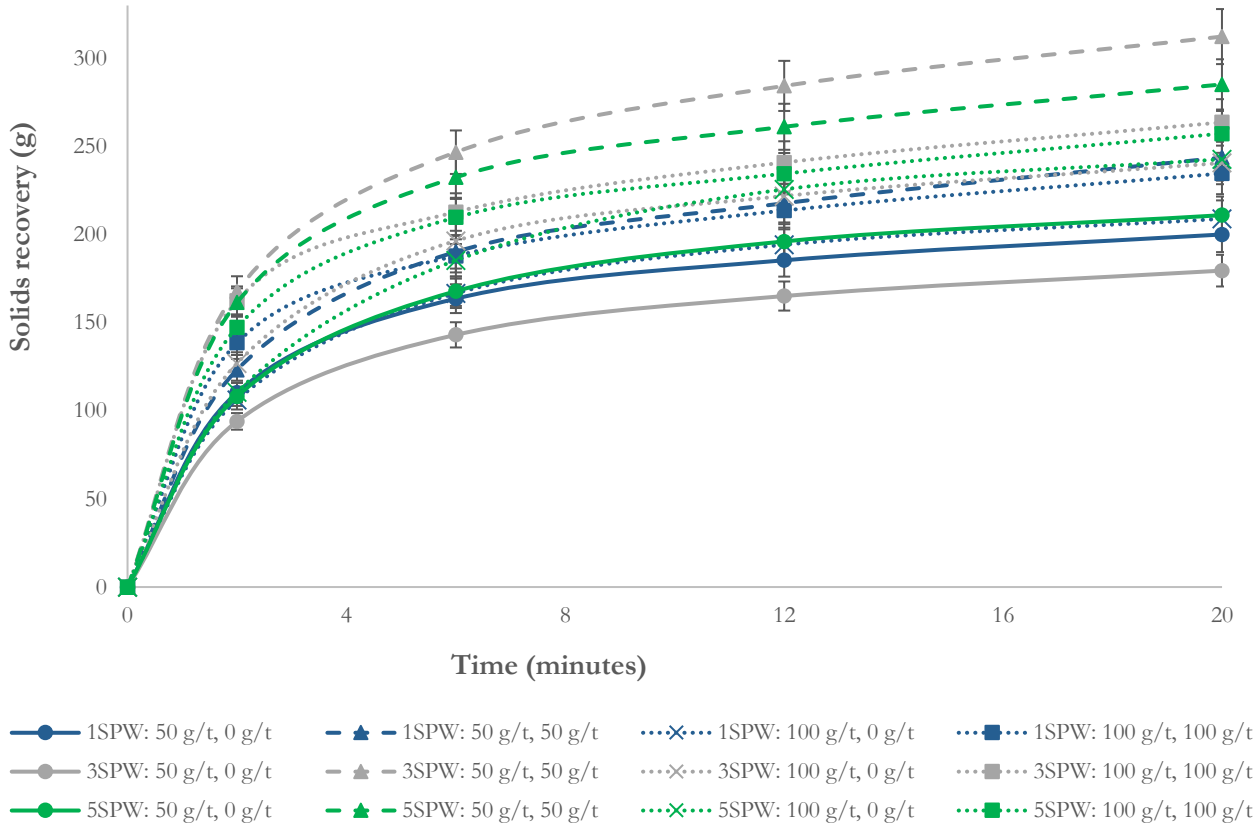

Figure 5.25: Solids recovery vs. time for Ore A; 50 and 100 g/t SIBX, in all once-recycled waters

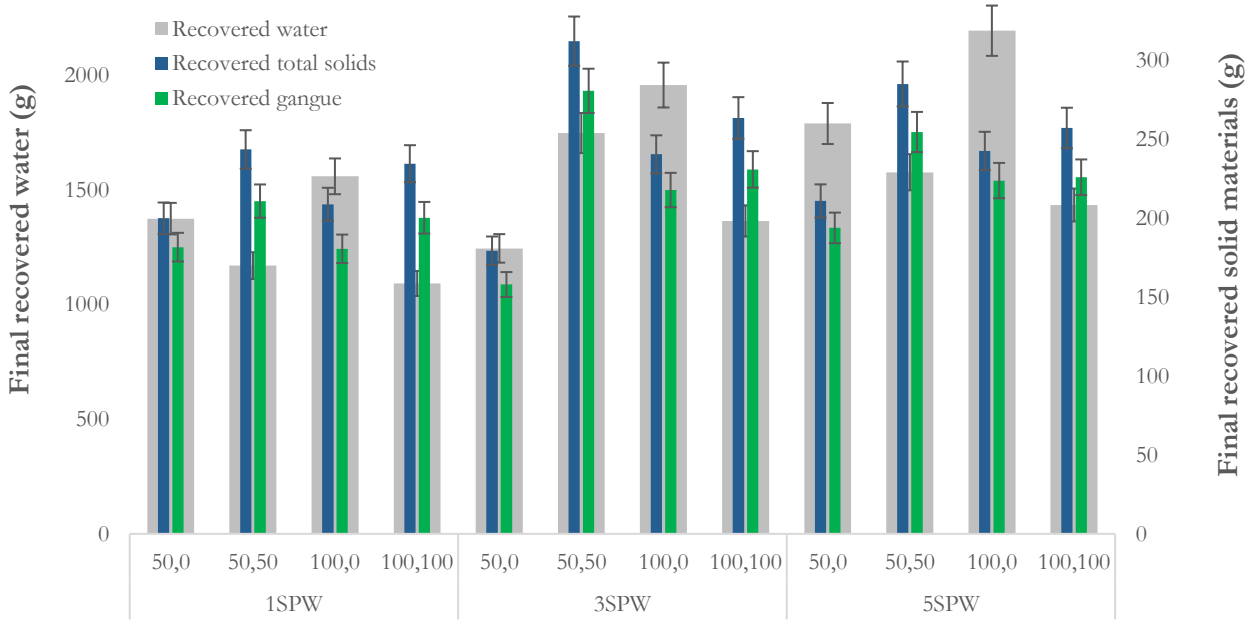

Figure 5.26: Total solids and gangue material vs. water recovered for Ore A; 50 and 100 g/t SIBX, in all once-recycled waters

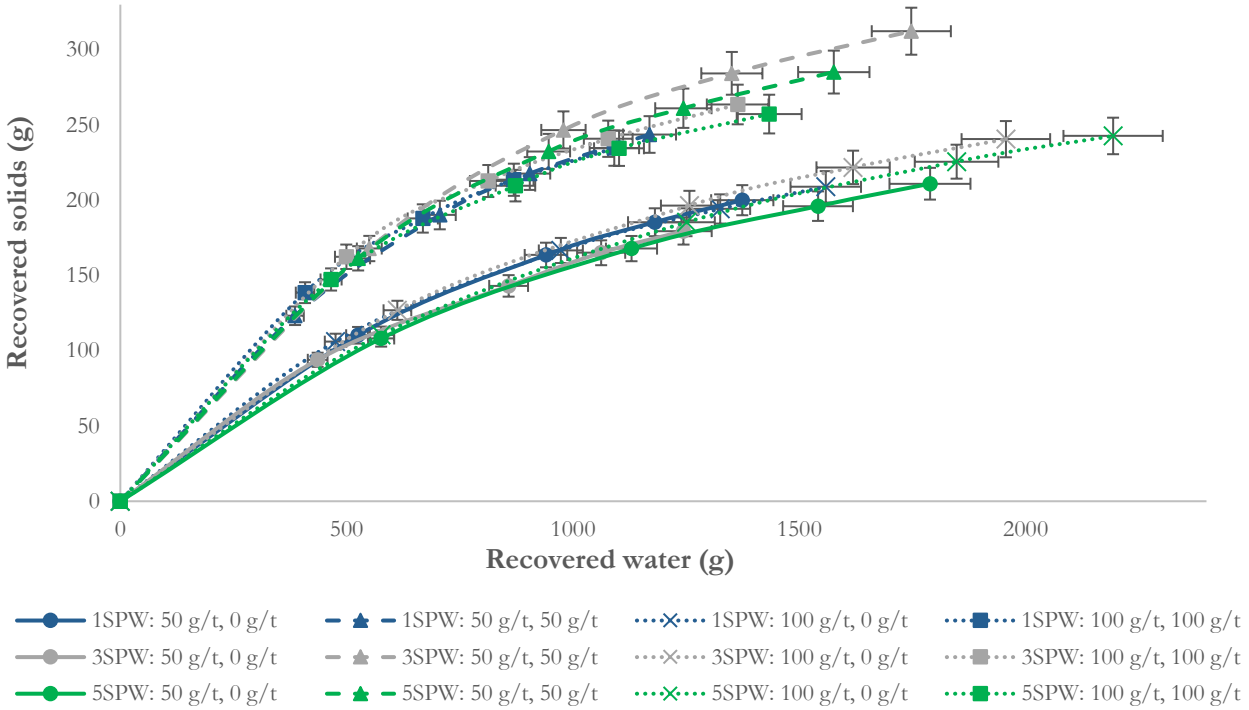

Figure 5.27: Solids vs. water recovery for Ore A; 50 and 100 g/t SIBX, in all once-recycled waters

Figure 5.28 shows the recovery of copper as a function of time for all Run 2 tests, and Figure 5.29 illustrates the copper grade vs. recovery curves.

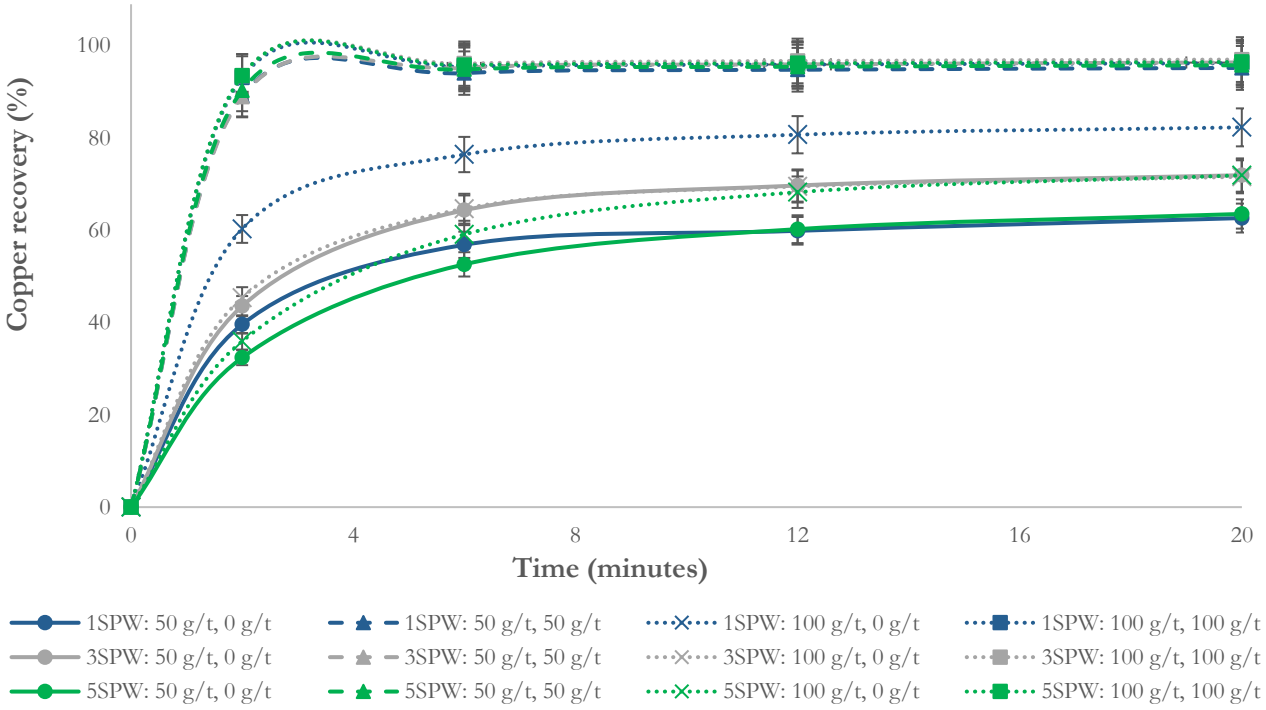

Figure 5.28: Copper recovery vs. time for Ore A; 50 and 100 g/t SIBX, in all once-recycled waters

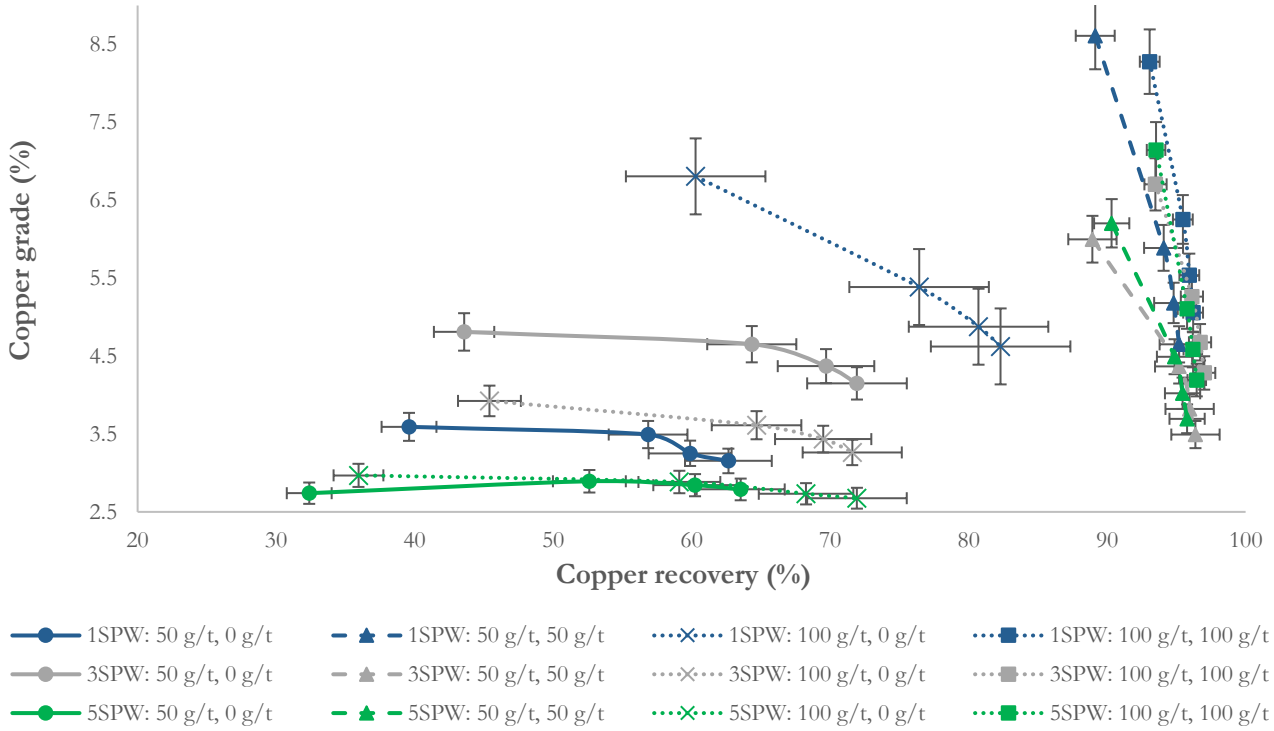

Figure 5.29: Copper grade vs. recovery for Ore A; 50 and 100 g/t SIBX, in all once-recycled waters

Like Run 1, the recycle runs with additional collector reported high recoveries, with the result being that the curves cluster to one extreme side (where the lowest final recovery was observed for *50,50* SIBX in 1SPW, at 95.2%; and the highest final recovery (97.0%) was for *100,100* SIBX in 3SPW). Thus, as with Run 1, the final values, especially the grades, are better presented in bar-form.

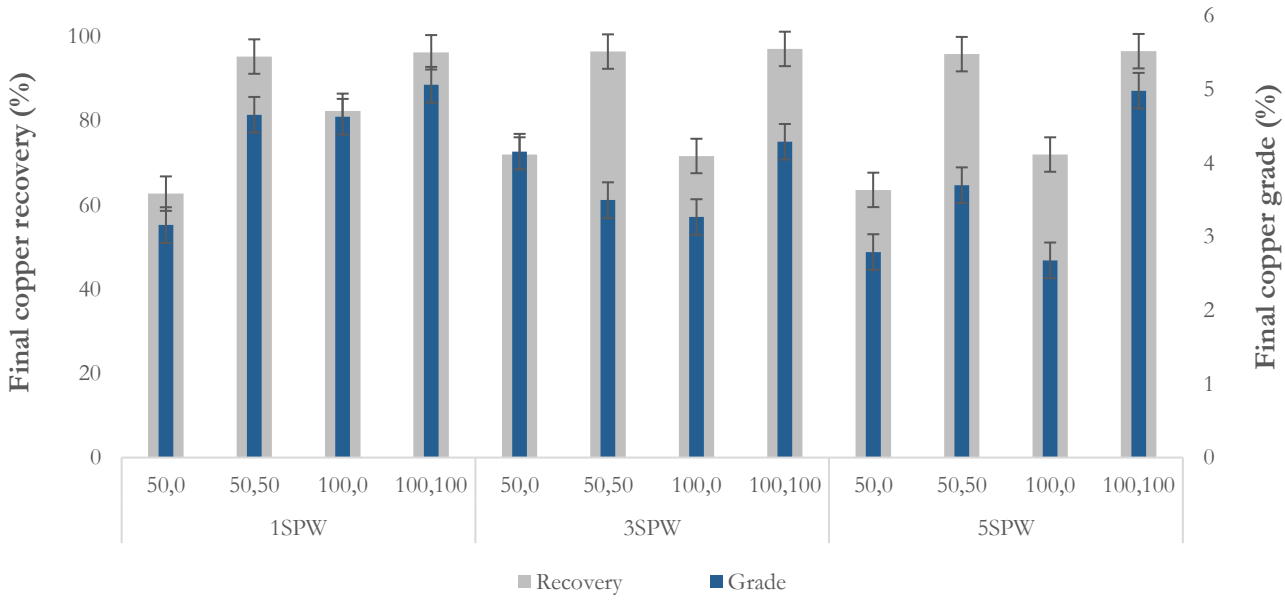

Figure 5.30: Final copper grade vs. recovery for Ore A; 50 and 100 g/t SIBX, in all once-recycled waters

Of the tests without additional collector in the recycle, *100,0* SIBX in 1SPW stands out as the best performer, reporting the highest final recovery (82.3%) for that batch of tests, and reporting a final grade of 4.62%. On the other hand, *50,0* SIBX in 1SPW reported the lowest final recovery (62.7%), while *50,0* and *100,0* SIBX in 5SPW stand out as the worst performers in terms of final grade, with each test reporting a value of approximately 2.70%. For the tests with additional collector in the recycle, and for both 50 and 100g/t SIBX, flotation in 1SPW yielded higher grades than flotation in the other waters. And overall, the grades yielded in these recycle runs were lower than those obtained in Run 1.

Figure 5.31 and Figure 5.32 show the SIBX concentration remaining in solution from C1 to the tailings; this, for a Run 2 re-dosage of 50 and 100 g/t SIBX, respectively.

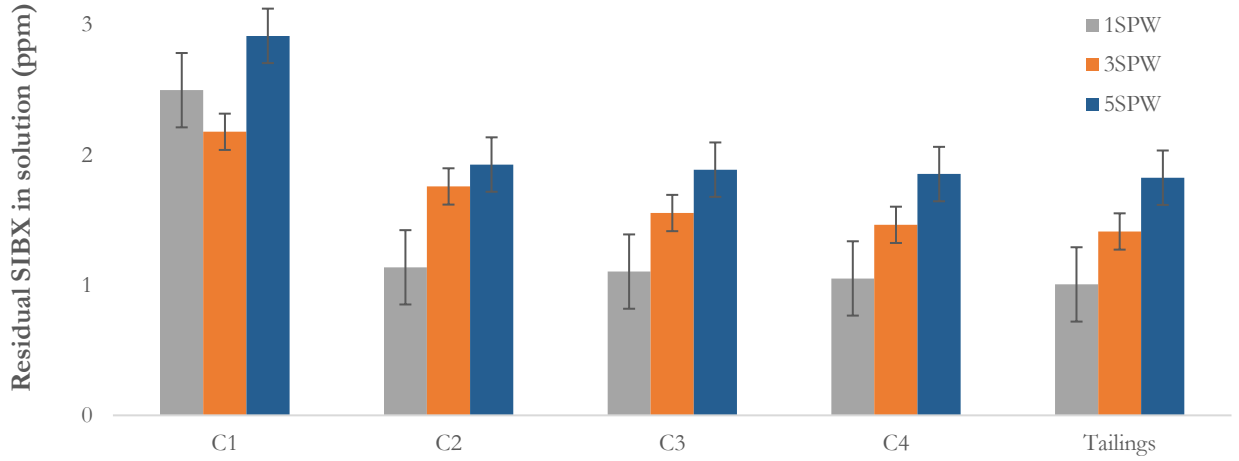

Figure 5.31: The collector concentration remaining in C1 to C4 for an initial dosage of 50 g/t SIBX in Run 2

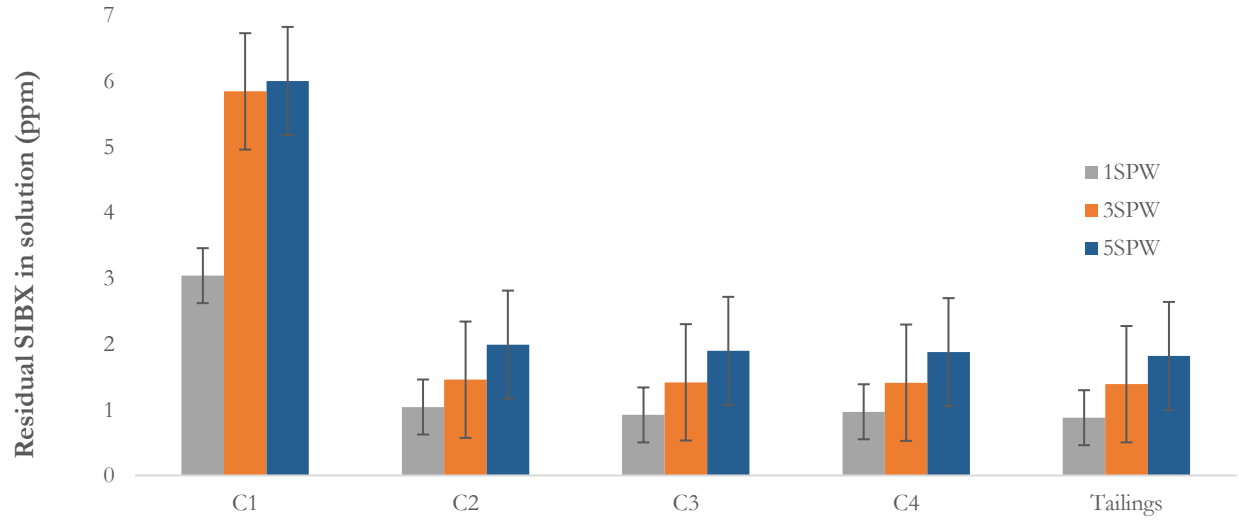

Figure 5.32: The collector concentration remaining in C1 to C4 for an initial dosage of 100 g/t SIBX in Run 2

For both dosages, and for all tested waters, the trend that was observed for Run 1 was repeated in that at the end of flotation the greater fraction of the dosed SIBX had been adsorbed. However, it can still be noted that the least amount of SIBX was adsorbed in 50 g/t dosage in 5SPW (where 90.1% of the initial collector was adsorbed and 1.82 ppm remained in solution) while the highest amount was adsorbed in 100 g/t dosage in 1SPW. For the *50,50* and *100,100* batch, Figure 5.33 summarises the residual SIBX concentration in C1 and the tailings of each water, and Figure 5.34 summarises the fraction of the initial concentration that was adsorbed for Run 2.

As with Run 1, the adsorption of SIBX across the different waters can be approximated by respective linear relationships. The R-squared values for the linear approximations are as follows: 1 for the 50 g/t dosage, and 0.998 for the 100 g/t dosage. Additionally, all Run 2 tests showed higher adsorption values than their Run 1 counterparts. For example, 30.7 ppm SIBX was adsorbed for a 100 g/t dosage in 5SPW-Run 1, while 33.8 ppm SIBX was adsorbed for the same dosage in 5SPW-Run 2.

And, for each water type in Run 1, between the 50 and 100 g/t dosages, a higher fraction of SIBX was adsorbed in the former case; but in Run 2, a greater fraction of the 100 g/t dosage was adsorbed. For example, in 5SPW-Run 1, for the 50 g/t SIBX dosage, 89.4% of the initial collector concentration was adsorbed, while for the 100 g/t dosage, 88.9% of the initial concentration was adsorbed. This is in contrast with observations made for 5SPW-Run 2, where, for the 50 g/t dosage, 90.1% of the initial collector concentration was adsorbed, while for the 100 g/t dosage, 94.9% of the initial concentration was adsorbed. A summary for all other comparisons is shown in Table 5.2.

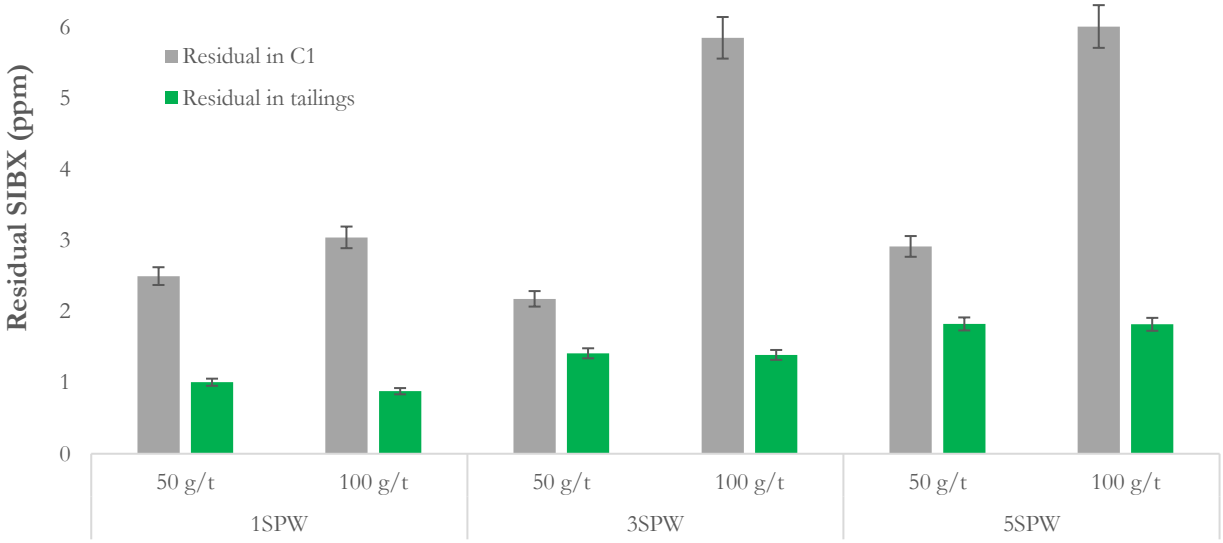

Figure 5.33: A summary of the amount of adsorbed SIBX in Run 2

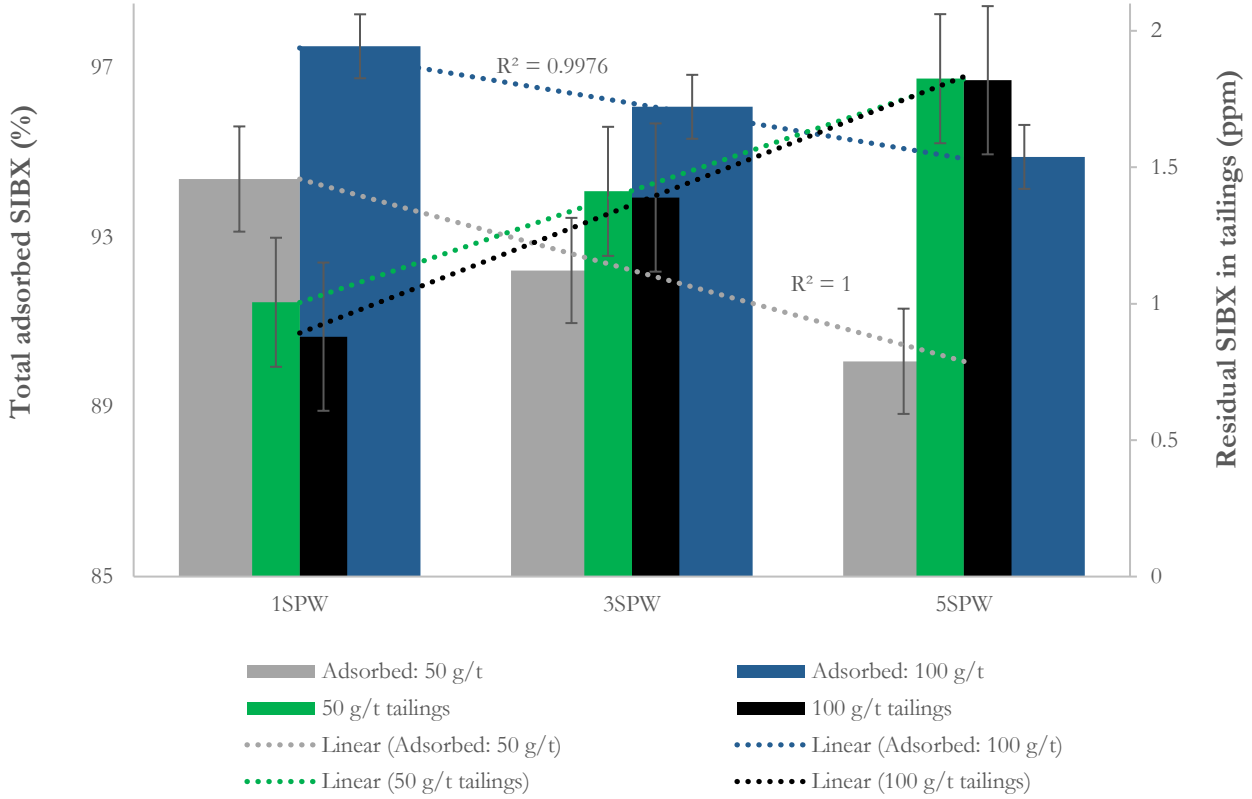

Figure 5.34: A summary of the amount of adsorbed SIBX in mg/litres (=ppm), plotted against the fraction of the initial dosage that was adsorbed in Run 2

Table 5.2: A comparison of residual SIBX concentrations in Run 1 and Run 2 (*50,50* and *100,100*)

Water type	**1SPW**		**3SPW**		**5SPW**	
SIBX dosage (g/t)	50	100	50	100	50	100
Run 1						
SIBX in feed (ppm)	17.2	34.5	17.2	34.5	17.2	34.5
Adsorbed SIBX (ppm)	16.2	32.3	15.8	31.5	15.4	30.7
Adsorbed SIBX (%)	94.4	93.7	91.6	91.2	89.4	88.9
Run 2						
SIBX in feed (ppm)	17.8	35.1	18.1	35.3	18.4	35.6
Adsorbed SIBX (ppm)	16.8	34.2	16.7	33.9	16.6	33.8
Adsorbed SIBX (%)	94.4	97.5	92.2	96.1	90.1	94.9

Figure 5.35 plots the amount of xanthate that was adsorbed against the final water recovery, and Figure 5.36 plots the total adsorbed xanthate against the total amount of recovered solids. For all tested waters, more water was recovered when less SIBX was adsorbed. In 1SPW and 5SPW, the difference in recovery is slight. In 3SPW, the difference is more pronounced; 1750 g water was recovered when 16.7 ppm SIBX was adsorbed, while 1360 g was recovered when 33.9 ppm SIBX was adsorbed.

Additionally, for all waters, more solids were recovered when less SIBX was adsorbed. And again, the difference in the recovery values was most pronounced for 3SPW and least pronounced for 1SPW.

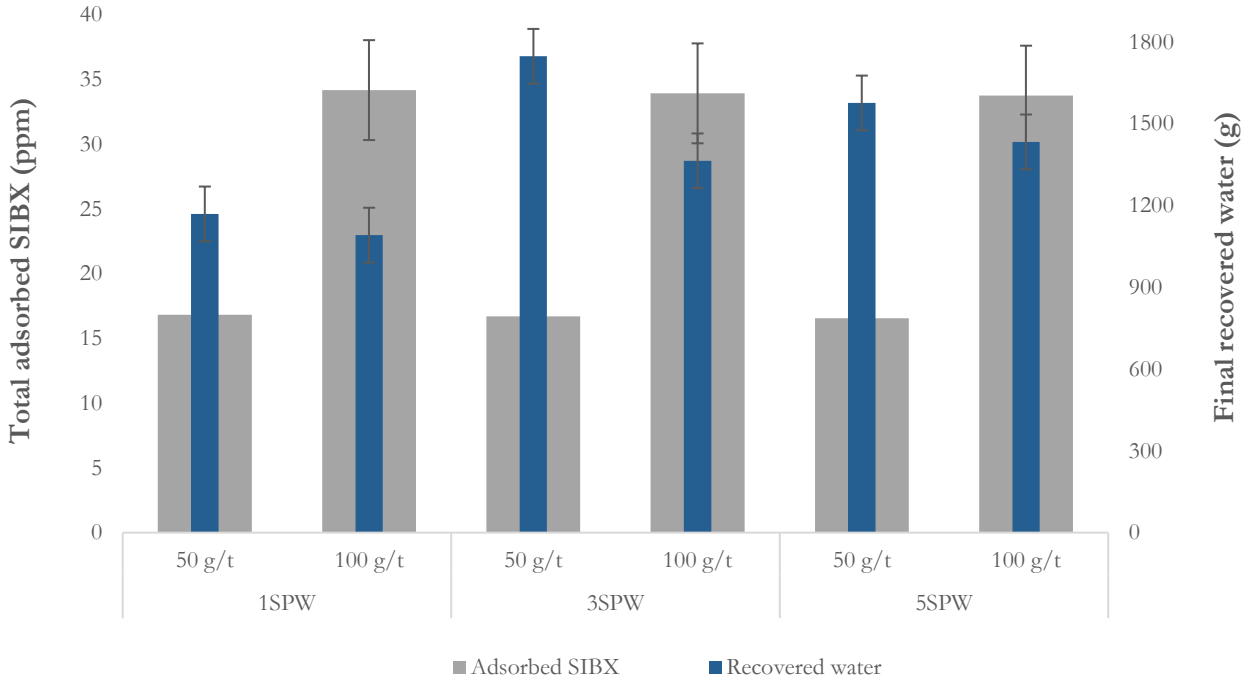

Figure 5.35: The total adsorbed SIBX plotted against final water recovery for all once-recycled waters

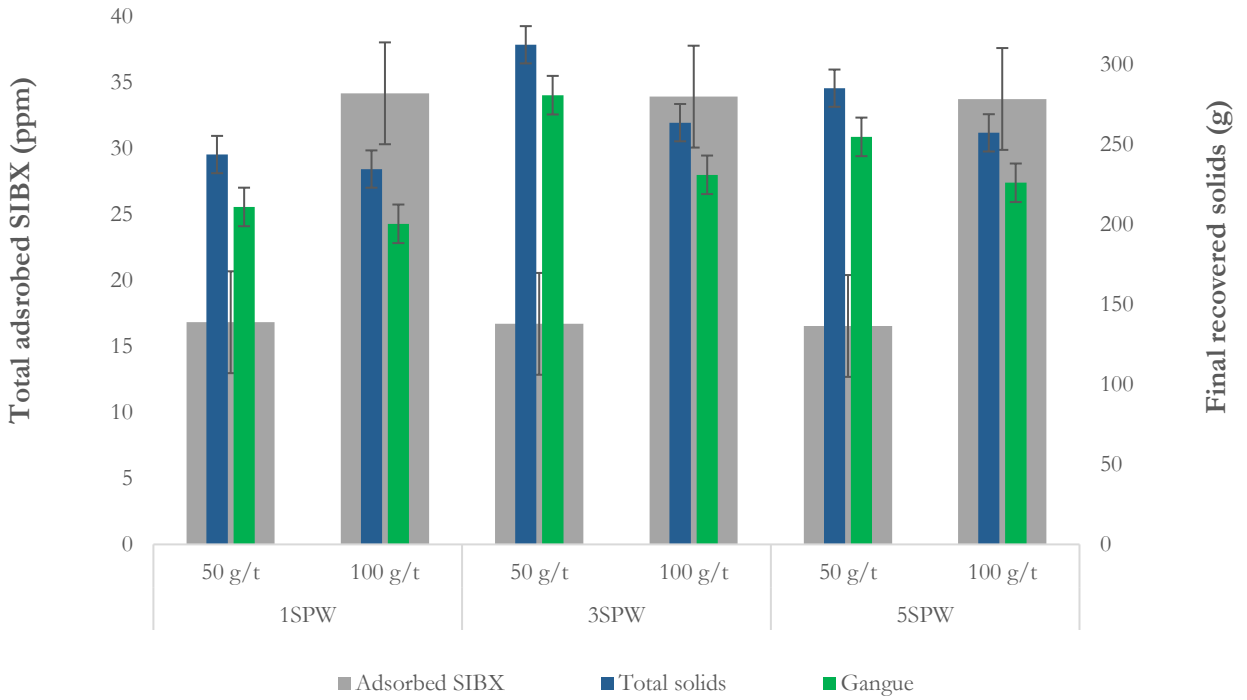

Figure 5.36: The total adsorbed SIBX plotted against final solids recovery for all once-recycled waters

Figure 5.37 shows the final copper recoveries against the total amount of adsorbed xanthate (for all second batch Run 2 tests). As was the case with Run 1, the degree of collector adsorption did not affect the final copper recovery.

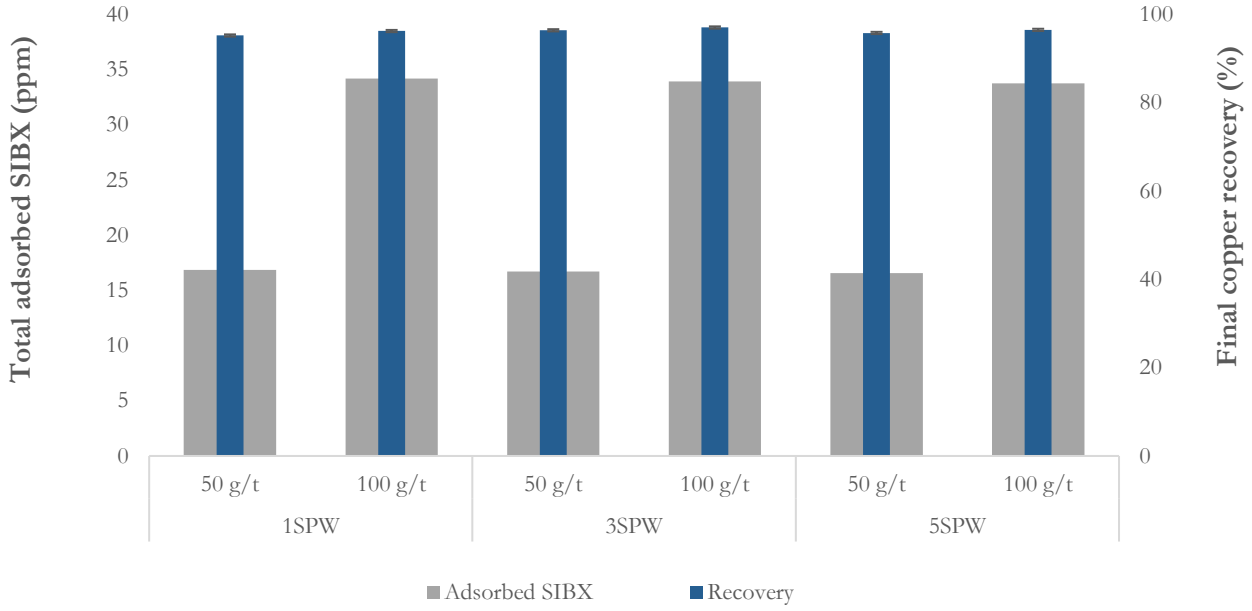

Figure 5.37: The total adsorbed SIBX plotted against final copper recovery for all tested waters in Run 2

Figure 5.38 shows the final copper grades against the total adsorbed SIBX. For 3SPW and 5SPW, as more xanthate was dosed, and as more xanthate was adsorbed, the concentrate grades improved. The same improvement is observed in 1SPW, but only slightly.

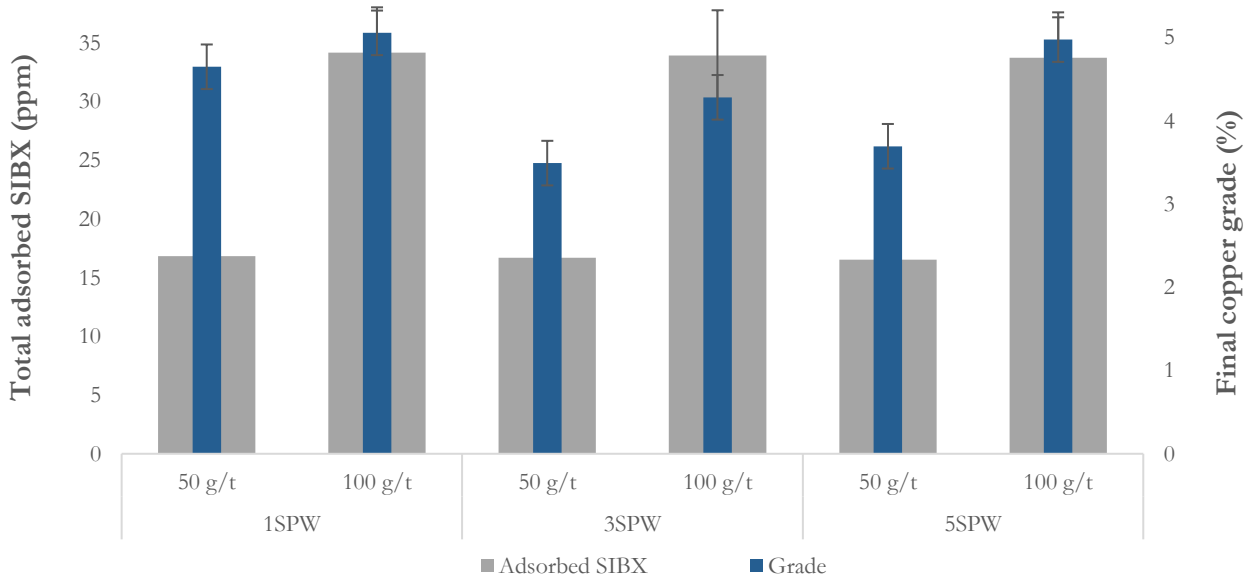

Figure 5.38: The total adsorbed SIBX plotted against final copper grade for all tested waters in Run 2

5.2.4. The Flotation Response of Ore A in Twice Recycled Water (Run 3)

The key notation that was used for Run 2 will also be applied for Run 3. That is to say: for recycle runs without additional collector, the key for the results is as follows: *50,0* for tests in which Run 1 was dosed with 50 g/t SIBX and Run 2 and Run 3 were not. For the recycle runs with additional collector, the key is thus: *50,50* for tests where Run 1, Run 2 and Run 3 were each dosed with 50 g/t SIBX. A similar system was employed for 100 g/t SIBX.

Since, as illustrated in Section 5.2.3., the least amount of collector was adsorbed in 5SPW-Run 2, while the most was adsorbed in 1SPW-Run 2, it was the case that flotation in 5SPW-Run 3 commenced with the most amount of SIBX when compared to the other waters, and flotation in 1SPW-Run 3 commenced with the least amount. Thus, for the different waters, Table 5.3 shows the collector concentrations at the beginning of Run 3.

Table 5.3: SIBX concentrations in the feed for the recycle runs (Run 3)

Water	1SPW	3SPW	5SPW
SIBX in *50,0* (ppm)	0.624	0.876	1.13
SIBX in *50,50* (ppm)	17.9	18.1	18.4
SIBX in *100,0* (ppm)	0.546	0.862	1.13
SIBX in *100,100* (ppm)	35.0	35.3	35.6

Figure 5.39 illustrates the effect of water quality on the recovery of water to the concentrate, and Figure 5.40 illustrates the effect of water quality on the recovery of solids. Figure 5.41 depicts the effect of water quality on the total recovery of water, solids and gangue materials; and Figure 5.42 depicts the recovery of solids to the concentrate as a function of water recovery.

For 1SPW and 5SPW, the *50,0* set of tests yielded more water and total solids recoveries than the *100,0*, while for 3SPW, the opposite is true.

Of the runs with additional collector in the recycle, the *50,50* and *100,100* sets yielded similar water and total solids recoveries when floated in 1SPW; in each case the recovered water mass was approximately 1160 g, the recovered total solids mass was approximately 245 g, and the recovered gangue material mass was approximately 210 g. On the other hand, when in 3SPW and 5SPW, the results of the same sets (*50,50* and *100,100*) were separated in that the *50,50* set yielded more recovered water, total solids, as well as gangue minerals.

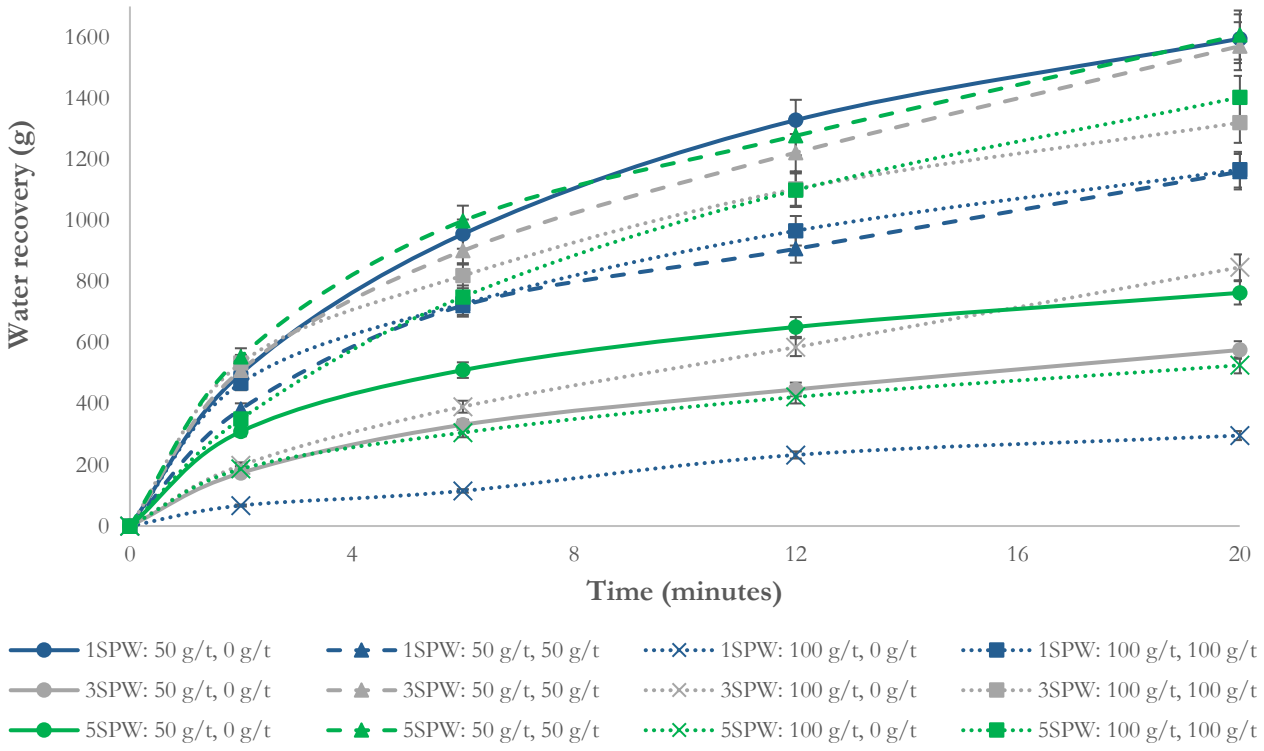

Figure 5.39: Water recovery vs. time for Ore A; 50 and 100 g/t SIBX, in all twice-recycled waters

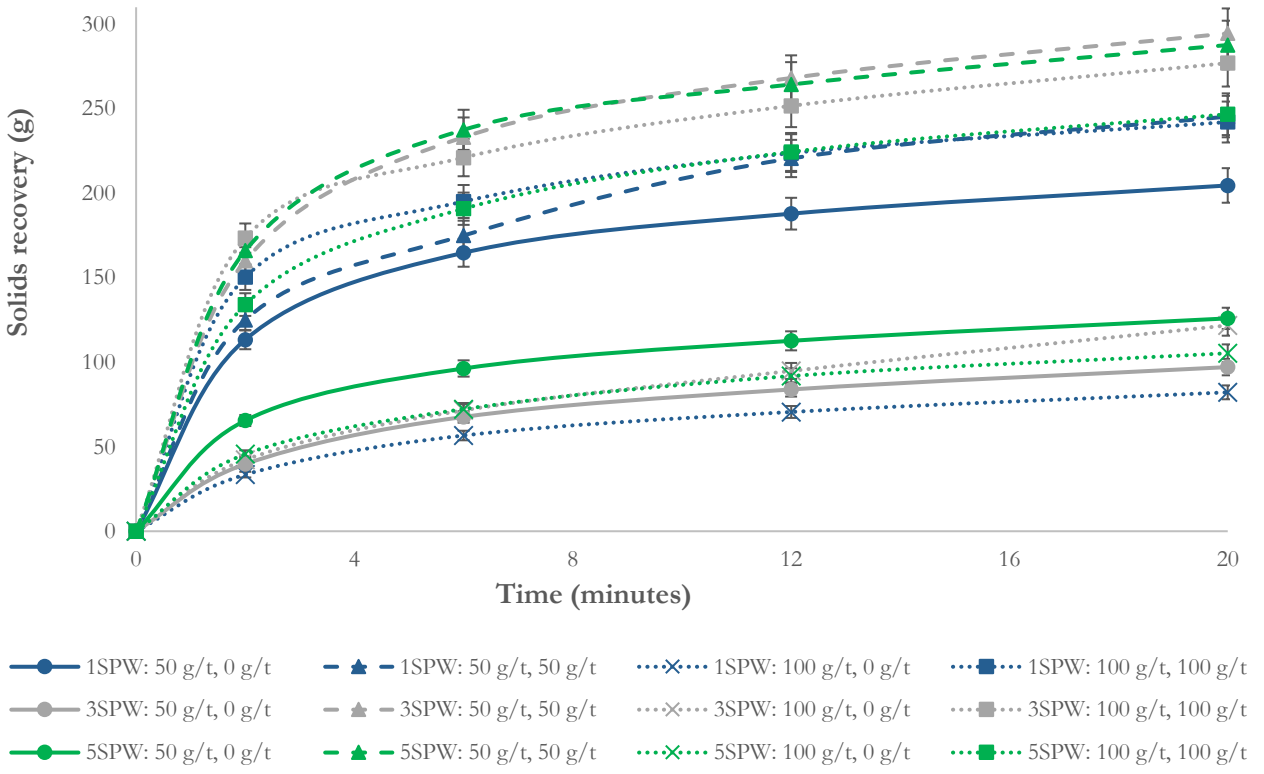

Figure 5.40: Solids recovery vs. time for Ore A; 50 and 100 g/t SIBX, in all twice-recycled waters

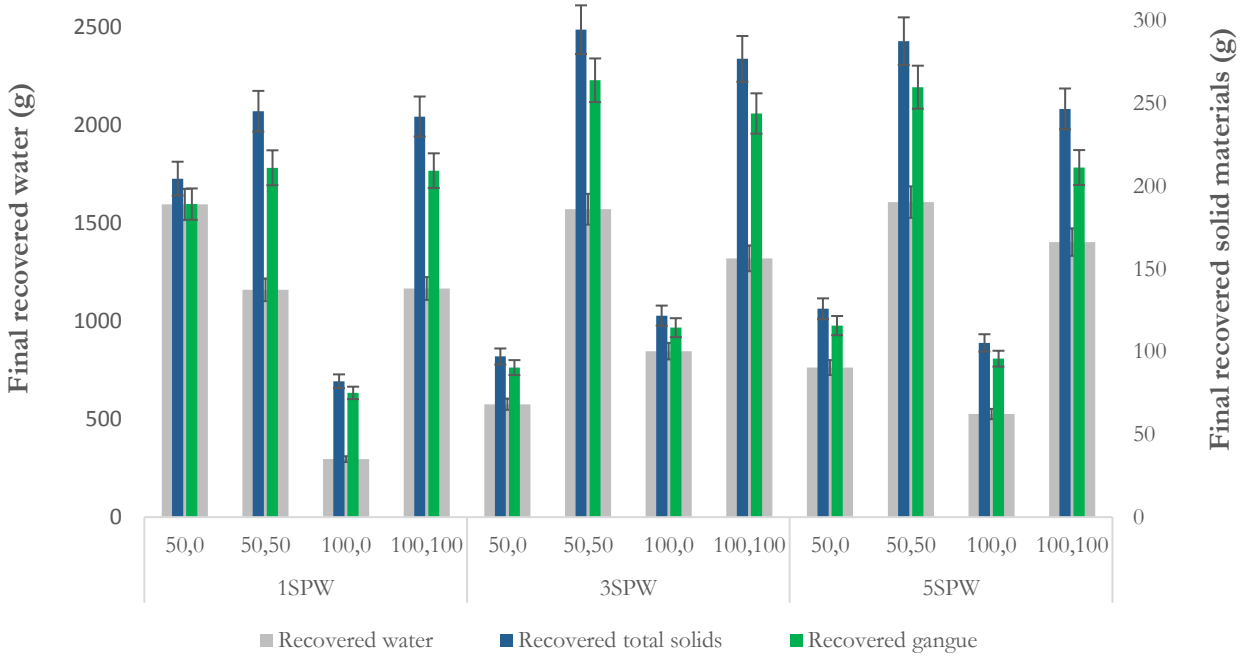

Figure 5.41: Total solids and gangue material vs. water recovered for Ore A; 50 and 100 g/t SIBX, in twice-recycled waters

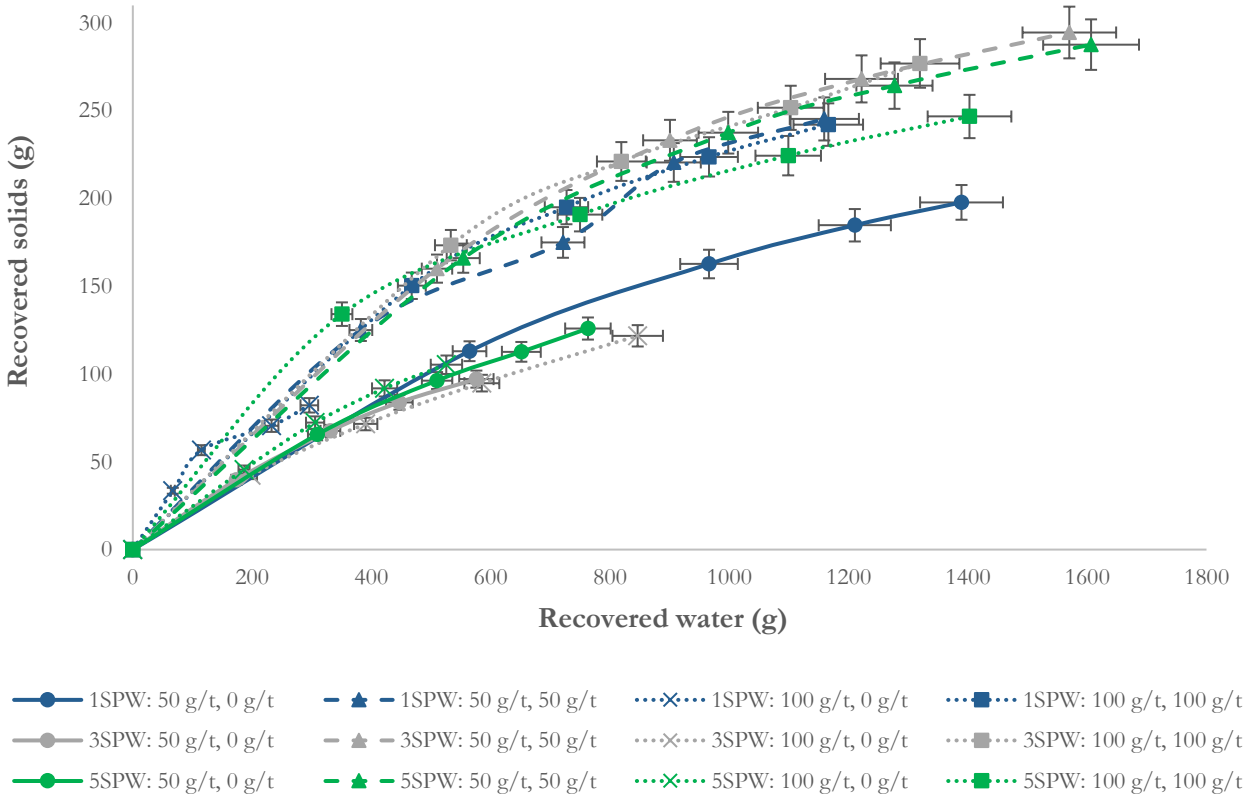

Figure 5.42: Solids vs. water recovery for Ore A; 50 and 100 g/t SIBX, in twice-recycled waters

Figure 5.43 illustrates the recovery of copper as a function of time for all Run 3 tests, and Figure 5.44 illustrates the copper grade vs. recovery curves. As with the preceding runs, the recycle runs with additional collector reported high recoveries, with the result being that the curves cluster to one extreme side (where the lowest final recovery was observed for *50,50* SIBX in 1SPW, at 95.7%; and the highest final recovery was observed for *100,100* SIBX in 3SPW, which reported a copper recovery of 97.4%). Thus, as with the preceding runs, the final values, especially the grades, are better represented in bar-form, by Figure 5.45.

Of the tests without additional collector in the recycle, *50,0* SIBX in 1SPW stands out as the best performer, reporting the highest final copper recovery (62.7%) for that batch of tests, and reporting a final grade of 2.59%. The best performer in terms of grade is *100,0* SIBX in 3SPW, which reported a final grade of 3.59%. On the other hand, *50,0* SIBX in 3SPW reported the lowest final copper recovery (18.6%) and the lowest final grade (2.39%).

For the tests with additional collector in the recycle, flotation in *100,100* SIBX in 5SPW yielded the highest grade (at 5.32%); this was followed by *50,50* and *100,100* SIBX in 1SPW (at 4.82% and 4.67%, respectively). For 1SPW and 3SPW, when compared to Run 2, the copper grade in Run 3 was improved in *50,50* SIBX but reduced in *100,100* SIBX; but for 5SPW, the reverse is true.

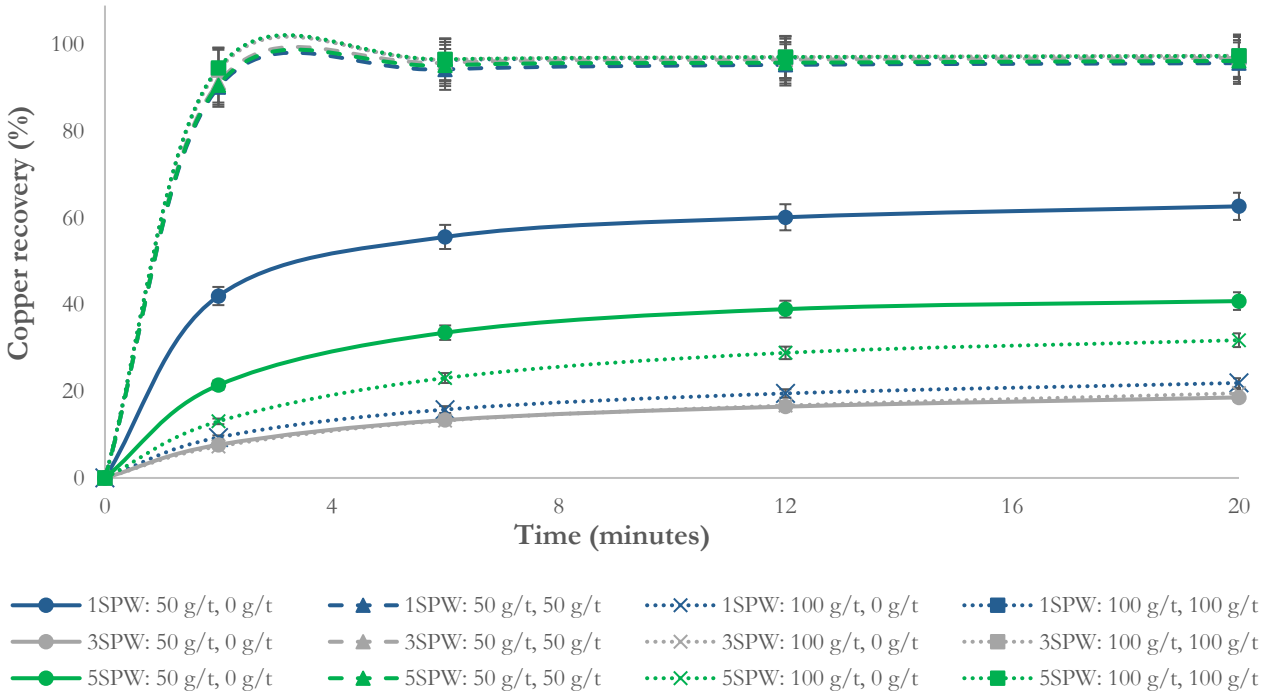

Figure 5.43: Copper recovery vs. time for Ore A; 50 and 100 g/t SIBX, in all twice-recycled waters

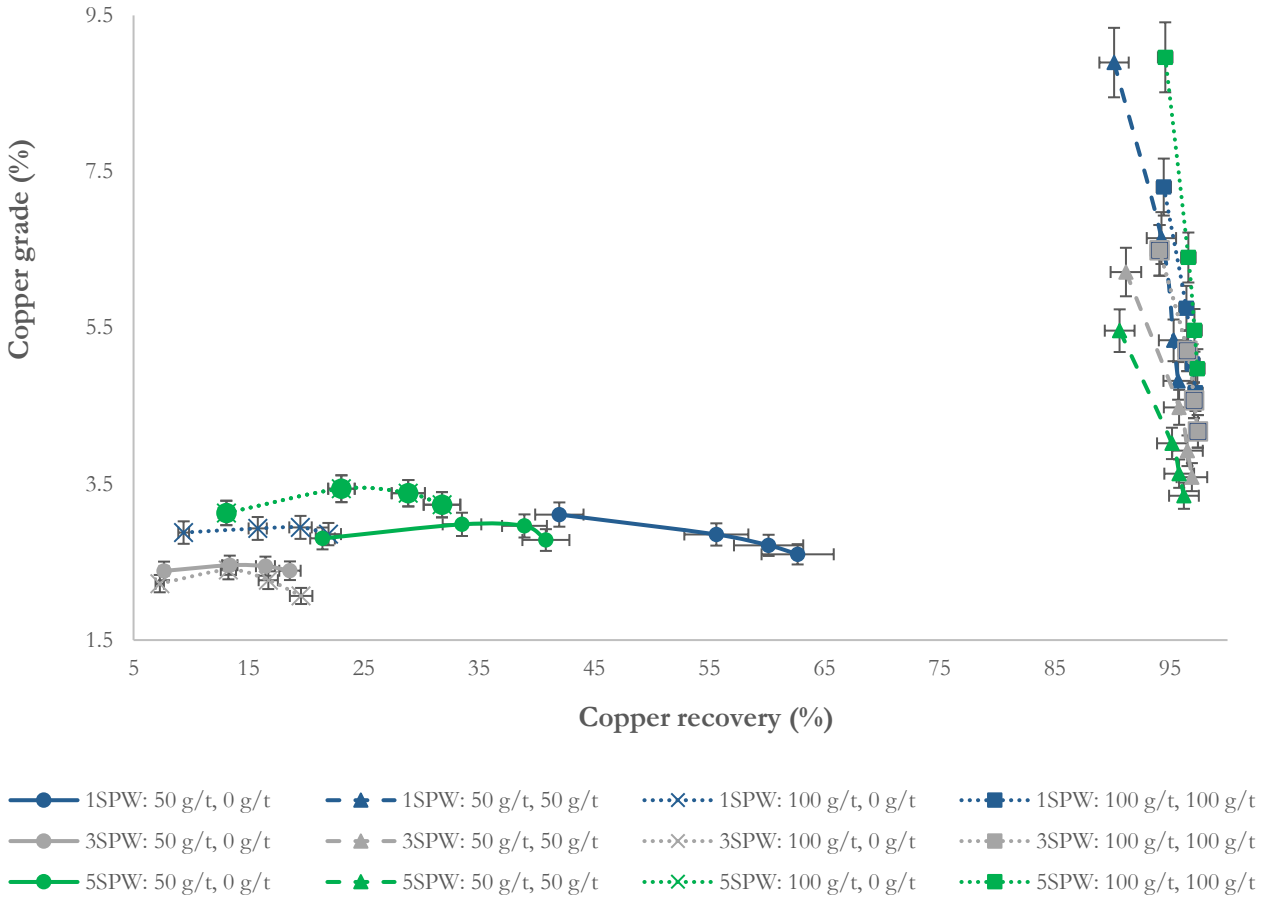

Figure 5.44: Copper grade vs. recovery for Ore A; 50 and 100 g/t SIBX, in all twice-recycled waters

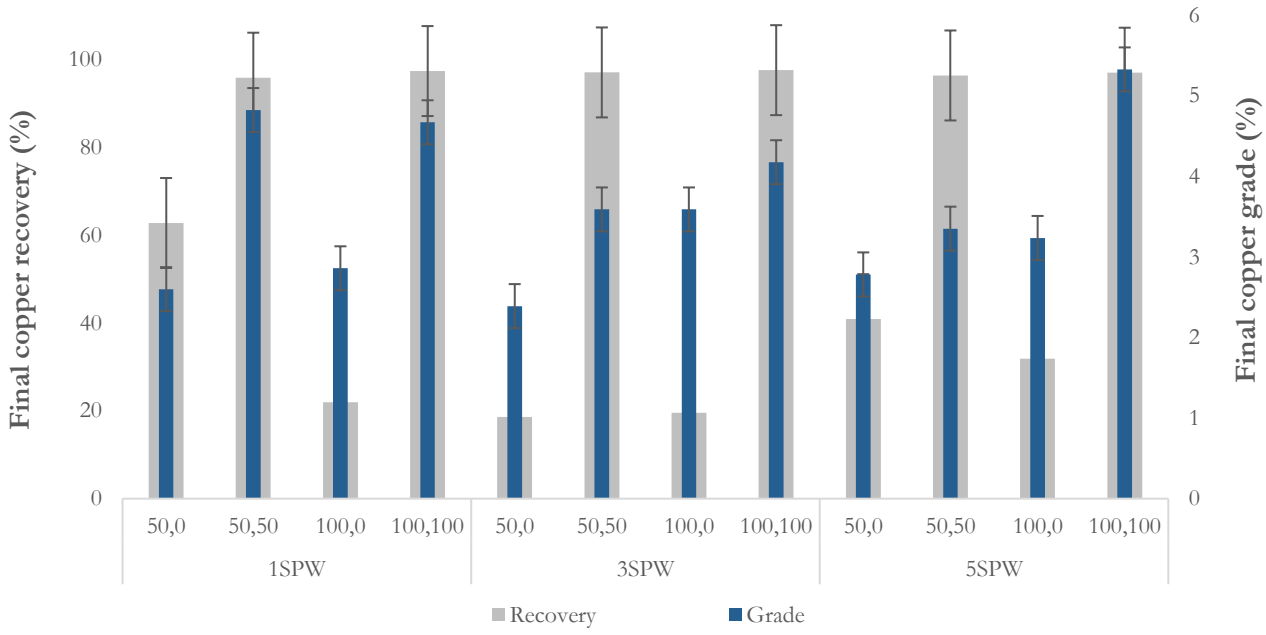

Figure 5.45: Final copper grade vs. recovery for Ore A; 50 and 100 g/t SIBX, in all twice-recycled waters

Figure 5.46 and Figure 5.47 show the SIBX concentration remaining in solution from C1 to the tailings; this, for a Run 3 re-dosage of 50 and 100 g/t SIBX, respectively. That is, for the *50,50* and *100,100* SIBX batch.

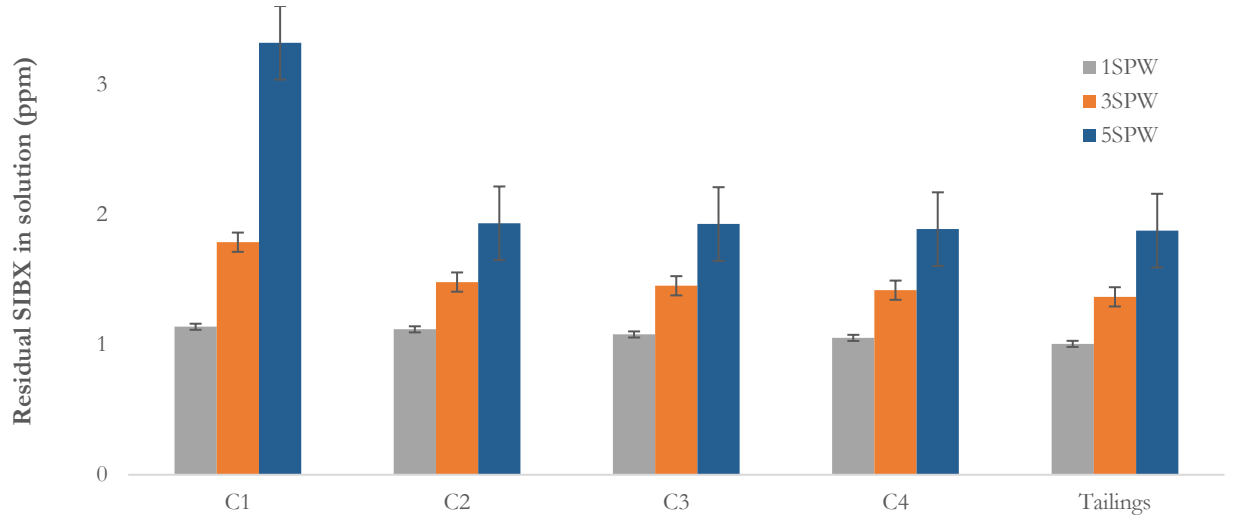

Figure 5.46:The collector concentration remaining in C1 to C4 for an initial dosage of 50 g/t SIBX in Run 3

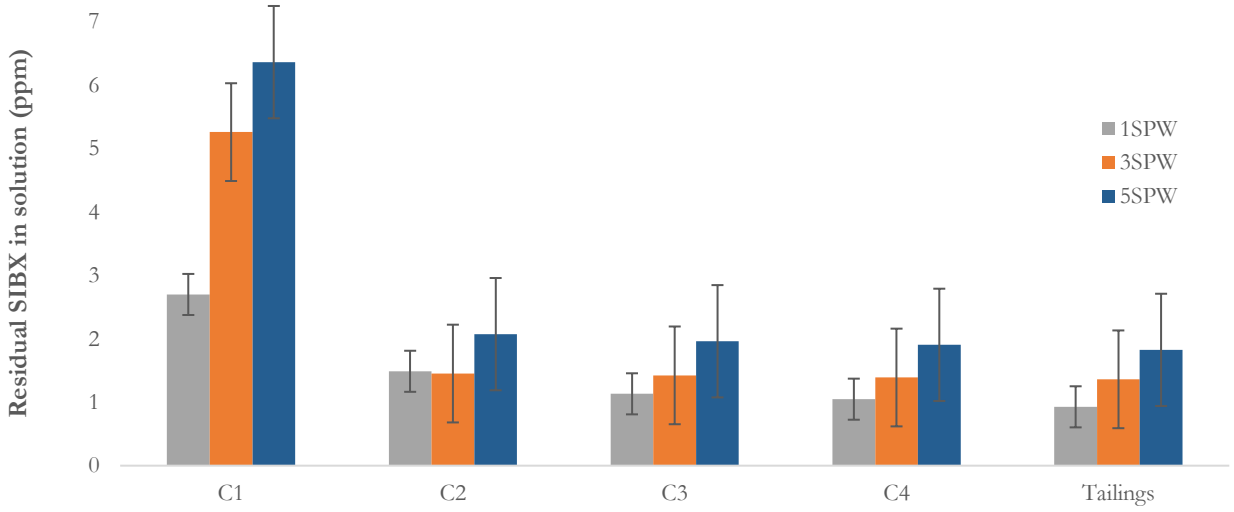

Figure 5.47: The collector concentration remaining in C1 to C4 for an initial dosage of 100 g/t SIBX in Run 3

Though, as with the preceding runs, the greater fraction of the dosed SIBX had been adsorbed at the end of flotation, it can still be noted that the least amount of SIBX was adsorbed in 50 g/t dosage in 5SPW (where 89.8% of the initial collector was adsorbed and 1.88 ppm remained in solution) while the most amount was adsorbed in 100 g/t dosage in 1SPW (where 97.4% of the initial collector was adsorbed and 0.927 ppm remained in solution).

For the *50,50* and *100,100* batch, Figure 5.48 summarises the residual SIBX concentration in C1 and the tailings of each water, and Figure 5.49 summarises the fraction of the initial concentration that was adsorbed for Run 3. As with the preceding runs, for the tested collector dosages, the adsorption of SIBX across the different waters can be approximated by respective linear relationships. The R-squared values for the linear approximations are as follows: 0.991 for the 50 g/t dosage, and 0.999 for the 100 g/t dosage.

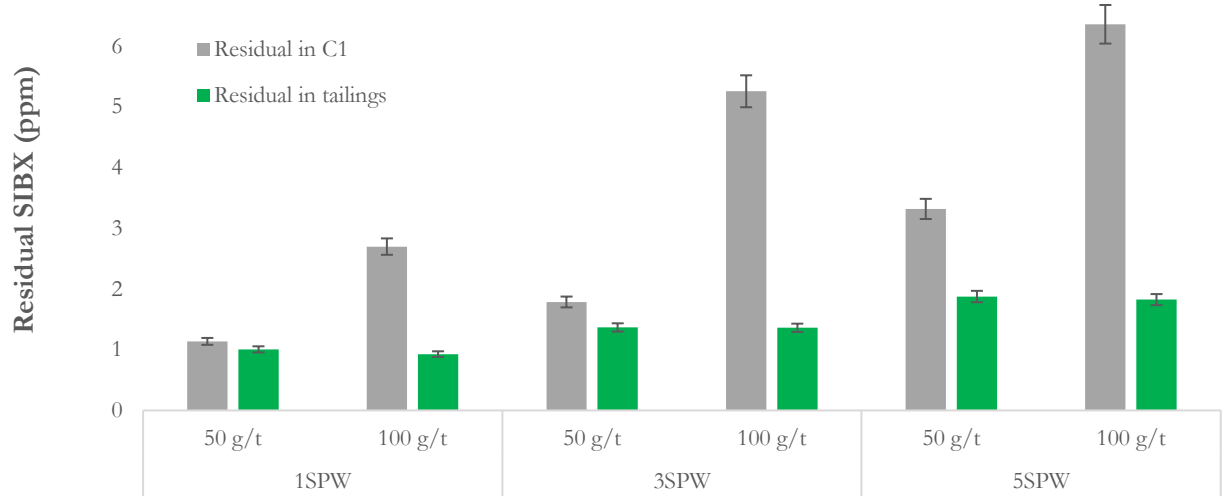

Figure 5.48: A summary of the amount of adsorbed SIBX in Run 3

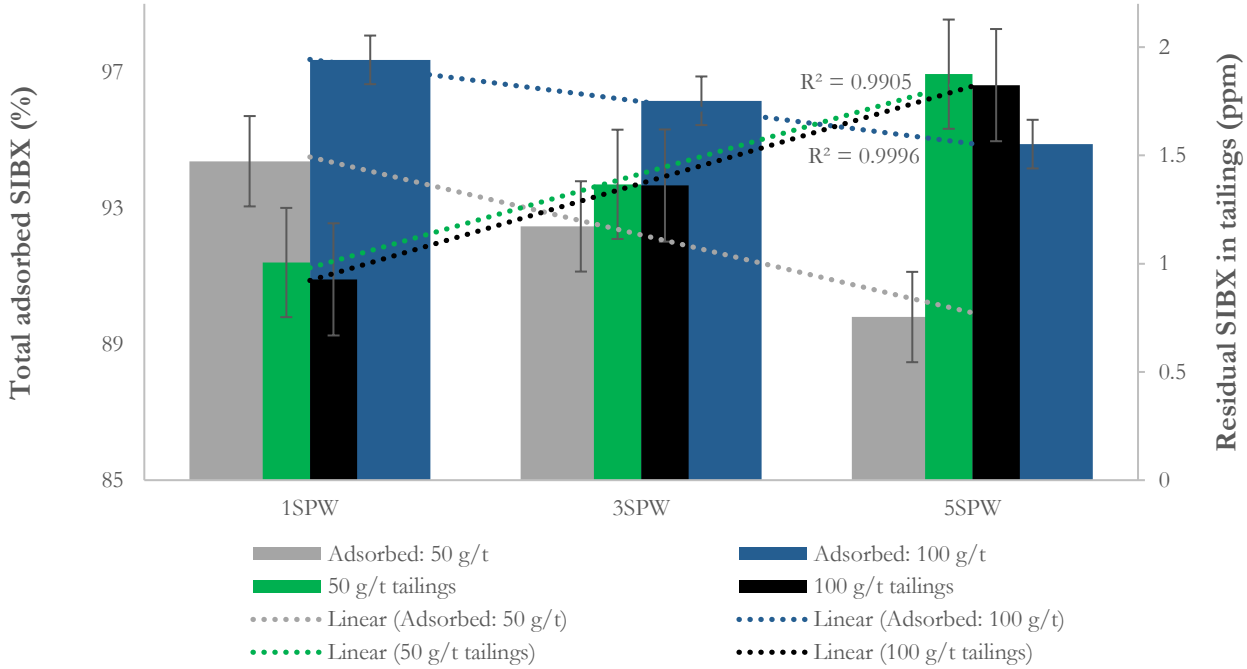

Figure 5.49: A summary of the amount of adsorbed SIBX in mg/litres (=ppm), plotted against the fraction of the initial dosage that was adsorbed in Run 3

All Run 3 tests showed higher adsorption values than their Run 1 counterparts. Additionally, while in Run 1 greater fractions of the 50 g/t xanthate dosage were adsorbed when compared to the adsorbed fractions for the 100 g/t dosage, the opposite was true for Run 3. A summary for all other comparisons is shown in Table 5.4.

Table 5.4: A comparison of residual SIBX concentrations in Run 1 and Run 3 (*50,50* and *100,100*)

Water type	**1SPW**		**3SPW**		**5SPW**	
SIBX dosage (g/t)	50	100	50	100	50	100
Run 1						
SIBX in feed (ppm)	17.2	34.5	17.2	34.5	17.2	34.5
Adsorbed SIBX (ppm)	16.2	32.3	15.8	31.5	15.4	30.7
Adsorbed SIBX (%)	94.4	93.7	91.6	91.2	89.4	88.9
Run 3						
SIBX in feed (ppm)	17.9	35.0	18.1	35.3	18.4	35.6
Adsorbed SIBX (ppm)	16.9	34.1	16.8	34.0	16.5	33.8
Adsorbed SIBX (%)	94.4	97.4	92.5	96.2	89.8	94.9

Figure 5.50 plots the amount of xanthate that was adsorbed against the final water recovery in Run 3, and Figure 5.51 plots the total adsorbed xanthate against the total amount of recovered solids. For 3SPW and 5SPW, more water was recovered when less SIBX was adsorbed. In 1SPW, there is little difference in the recoveries. For all waters, more solids were recovered when less SIBX was adsorbed. However, the difference in the recovery values was most pronounced for 5SPW and least pronounced for 1SPW.

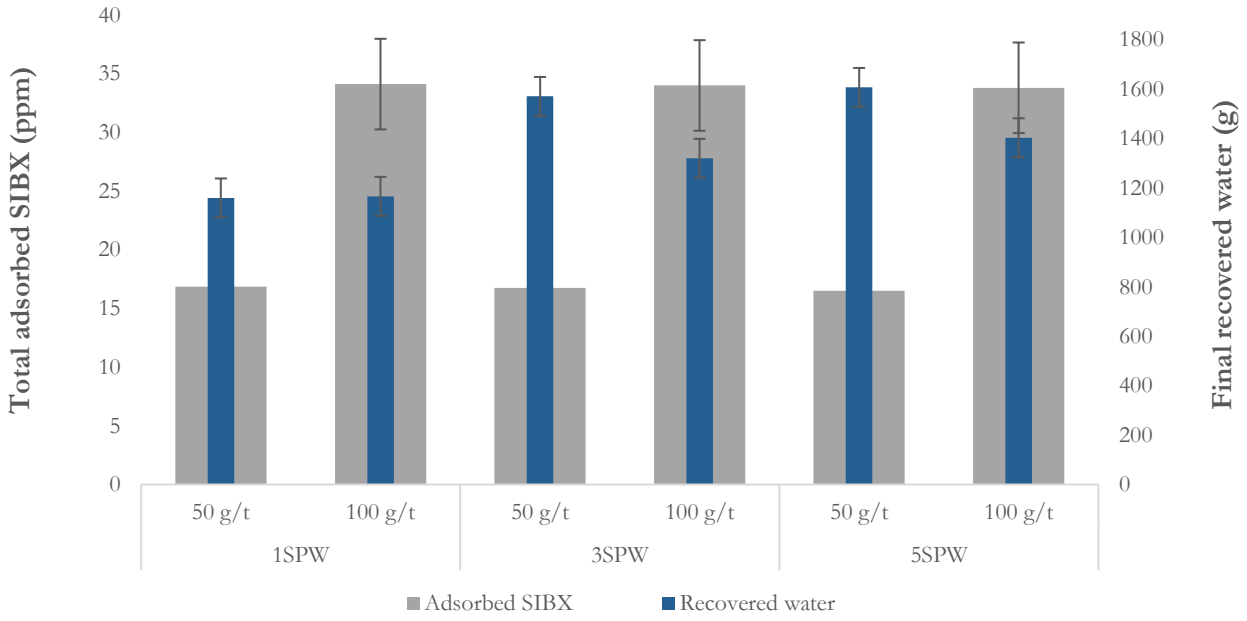

Figure 5.50: The total adsorbed SIBX plotted against final water recovery for all twice-recycled waters

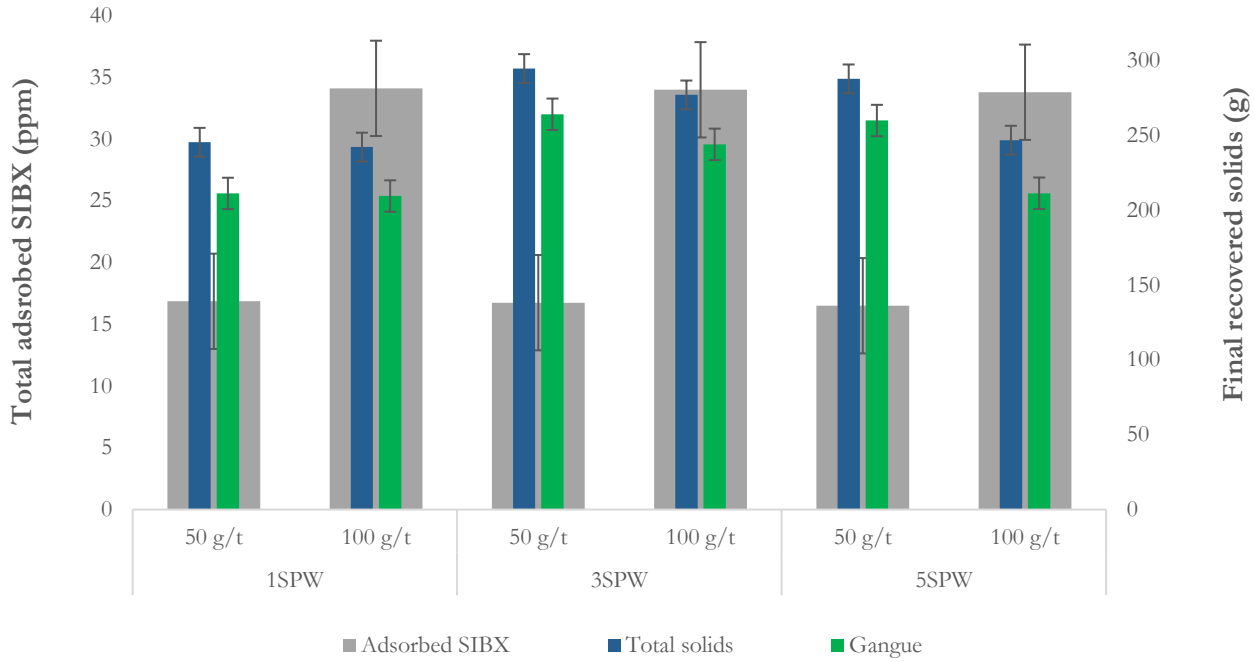

Figure 5.51: The total adsorbed SIBX plotted against final water recovery for all twice-recycled waters

Figure 5.52 shows the final copper recoveries against the total amount of adsorbed xanthate (for all second batch Run 3 tests). As was the case with the preceding runs, the degree of collector adsorption did not affect the final recovery.

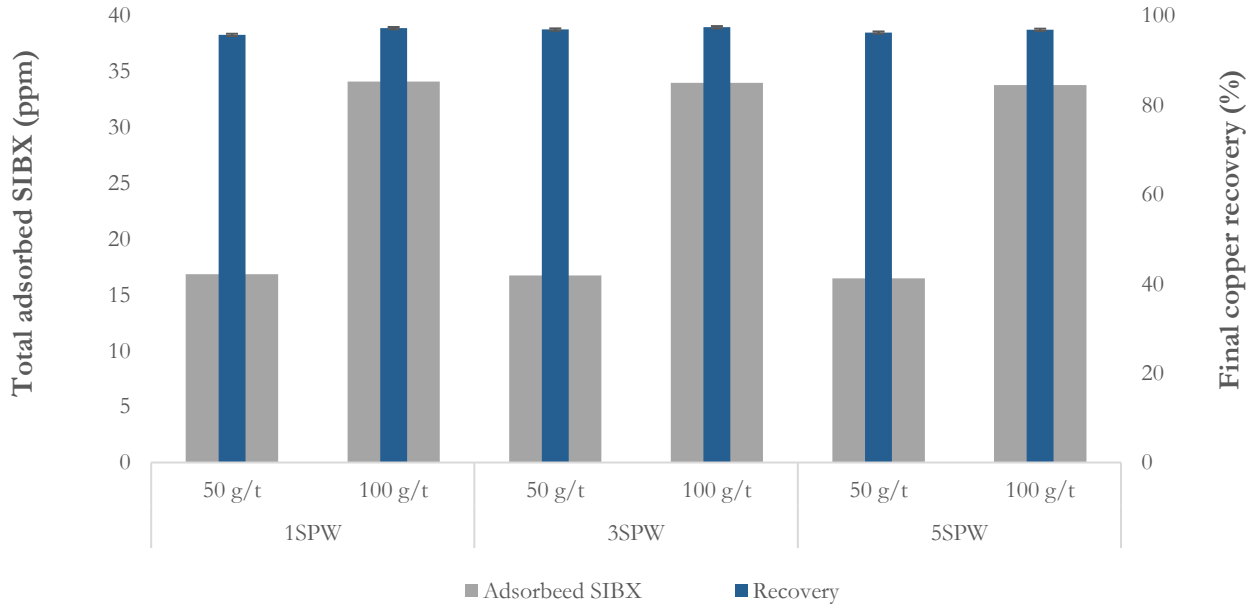

Figure 5.52: The total adsorbed SIBX plotted against final copper recovery for all tested waters in Run 3

Figure 5.53 shows the final copper grades against the total amount of adsorbed xanthate (for all second batch Run 3 tests). For 5SPW, as more xanthate was dosed, and as more xanthate was adsorbed, the concentrate grade improved more sharply than in the case of 3SPW. On the other hand, for 1SPW, there was a slight reduction in the copper grade as more xanthate was dosed, and as more xanthate was adsorbed (from 5.81% in 50 g/t SIBX to 4.67% in 100 g/t SIBX).

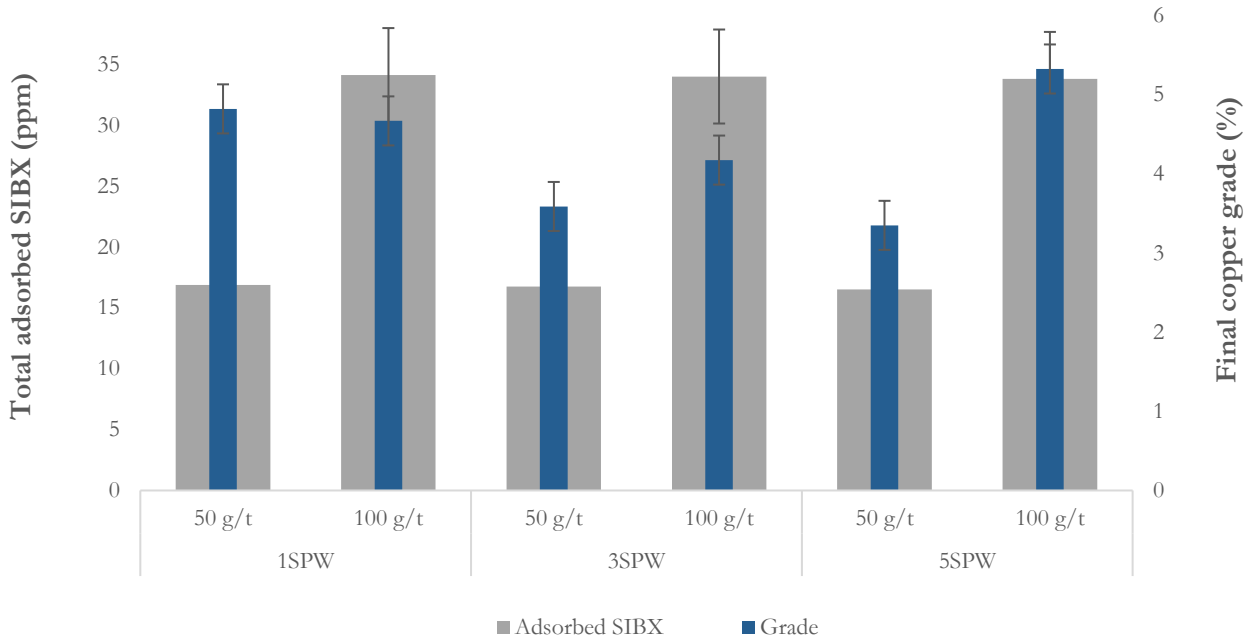

Figure 5.53: The total adsorbed SIBX plotted against final copper grade for all tested waters in Run 3

5.3. The Effects of Water Quality on the Flotation Response of Ore B

The effect of water quality on the flotation response of Ore B was tested at three levels of collector dosage i.e. with no collector; with 5 g/t aerophine and 100 g/t SIPX; and with residual collector, additional 5 g/t aerophine and additional 100 g/t SIPX. As a result of sequentially adding aerophine and SIPX, and as a result of redosing the system with frother after nine minutes of flotation, the grade-recovery and recovery-time curves show a bump at the 9-minute time interval. The test conditions (with regard to collector dosages) are summarised in Figure 5.54.

	Baseline	Additional SIPX dosed in recycle runs
Run 1	0 g/t	100 g/t
Run 2	-	Additional 100 g/t

Figure 5.54: The collector dosages tested at different water qualities for Ore B

5.3.1. The Flotation Response of Ore B in Collectorless Water

For flotation without added collector, Figure 5.55 and Figure 5.56 illustrate the respective effects of water quality on the recovery of water and the recovery of solids to the concentrate. Figure 5.57 depicts the effect of water quality on the recovery of water, total solids and gangue materials to the concentrate.

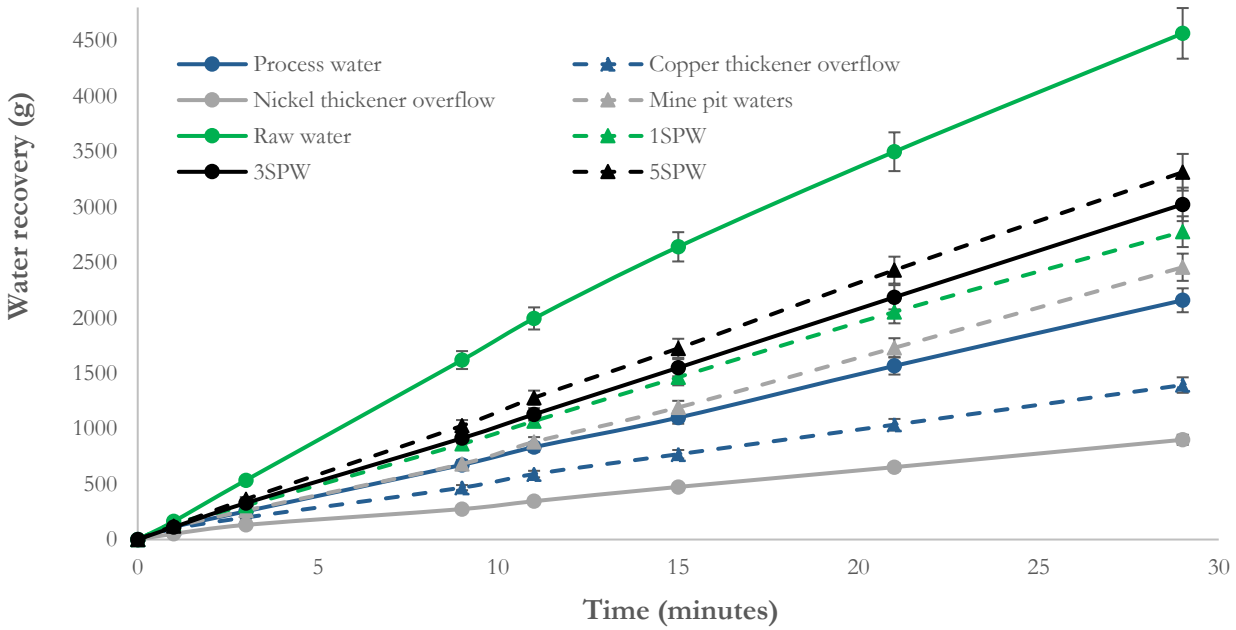

Figure 5.55: Water recovery vs. time for Ore B; no added collector and all tested waters

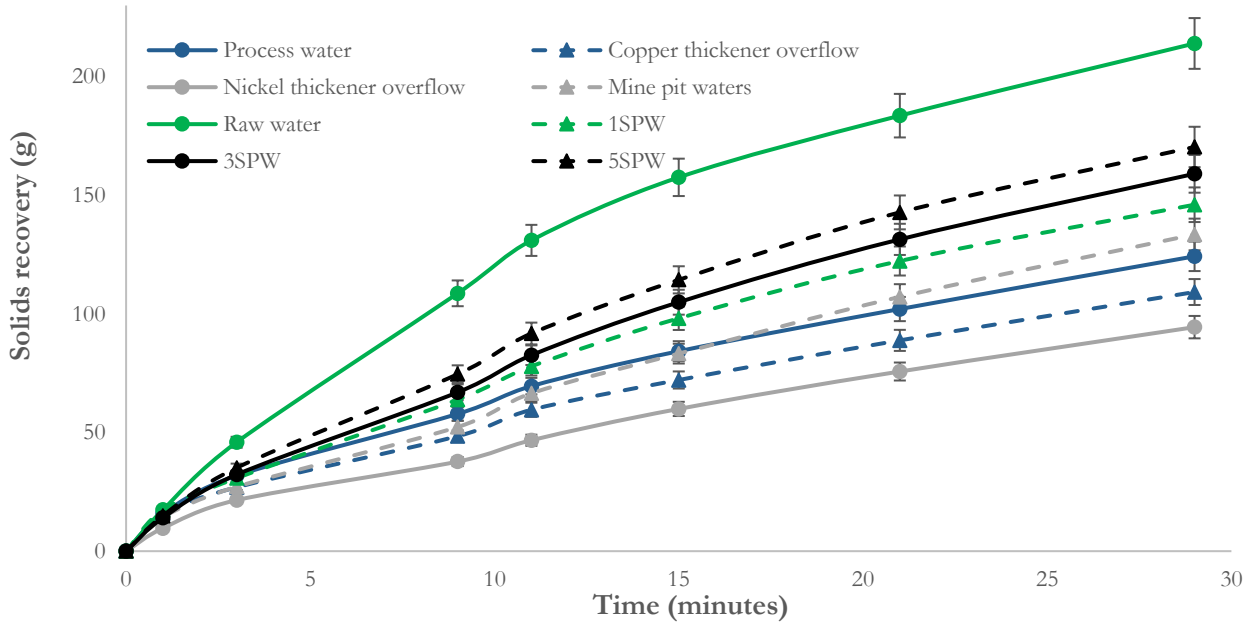

Figure 5.56: Solids recovery vs. time for Ore B; no added collector and all tested waters

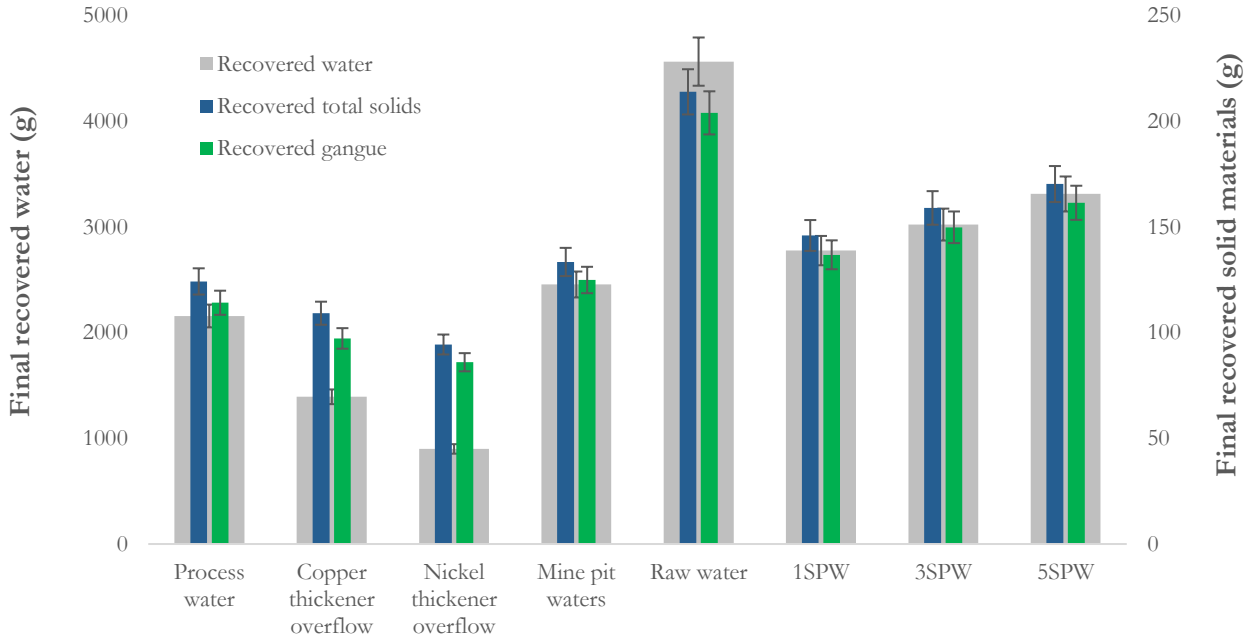

Figure 5.57: Total solids and gangue material vs. water recovered for Ore B; no added collector and all tested waters

The observed trend is that as the water recovery increased, so did the total mass of solids (and thus gangue minerals). The mass of total solids and water recovered in raw water flotation were nearly twice as much as the recoveries in the other waters. The synthetic plant waters exhibited an expected trend in that, as the ionic strength increased, so did the total amounts of the recovered water and solids.

This is further highlighted by Figure 5.58, which depicts the recovery of solids to the concentrate as a function of water recovery.

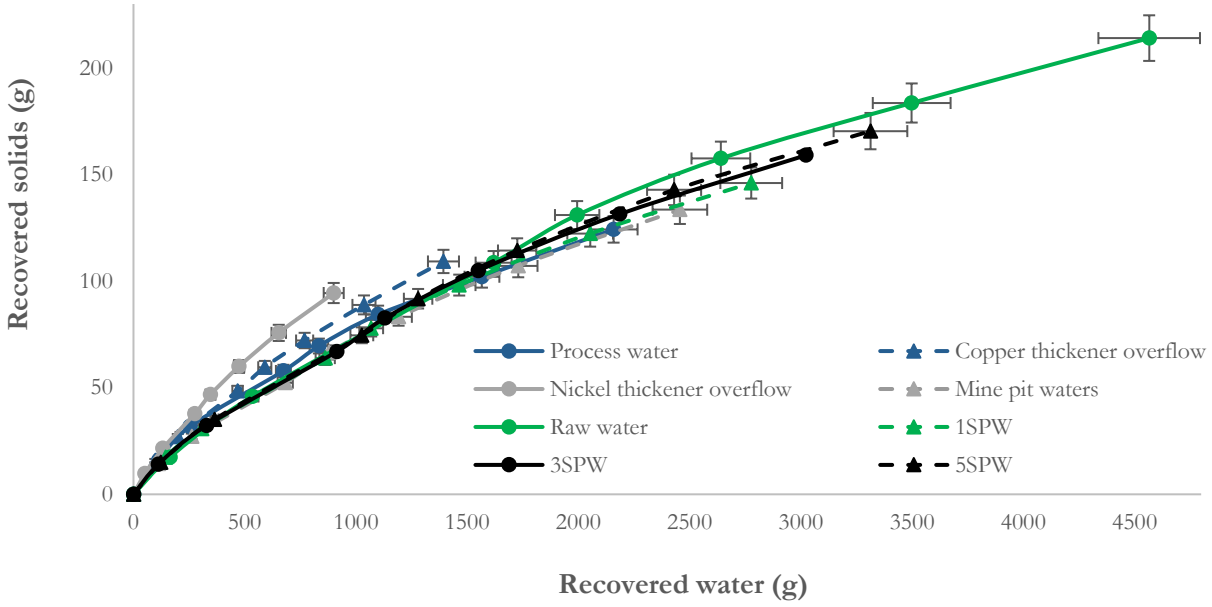

Figure 5.58: Solids vs. water recovery for Ore B; no added collector and all tested waters

Figure 5.59 and Figure 5.61 show the respective, collectorless recovery of copper and the collectorless recovery of nickel to the concentrate as a function of time. Figure 5.60 and Figure 5.62 illustrate the grade vs. recovery curves for copper and nickel, respectively.

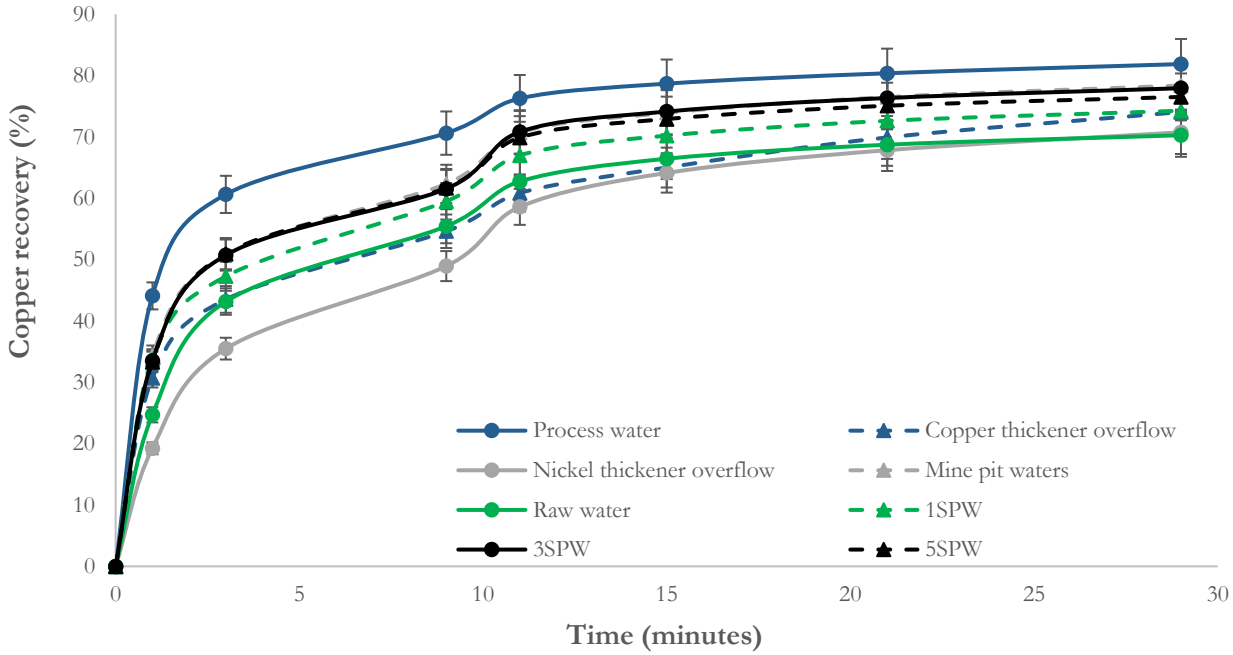

Figure 5.59: Copper recovery vs. time for Ore B; no added collector and all tested waters

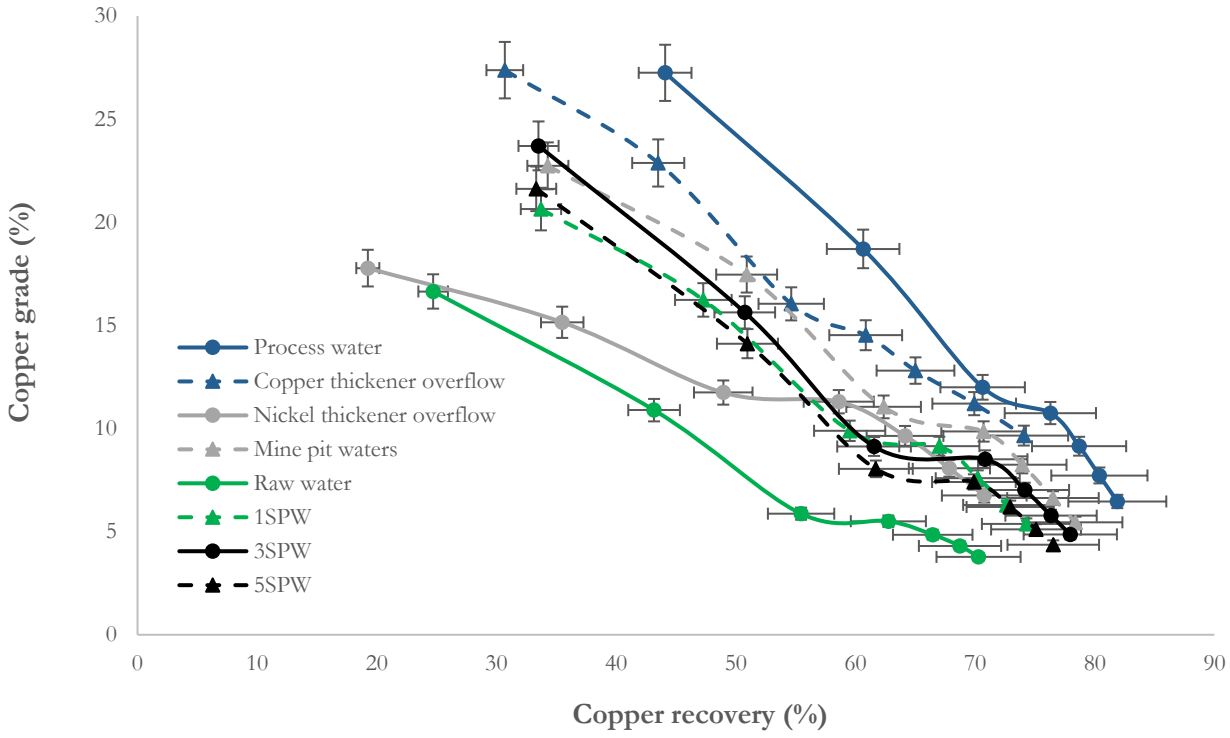

Figure 5.60: Copper grade vs. recovery for Ore B; no added collector and all tested waters

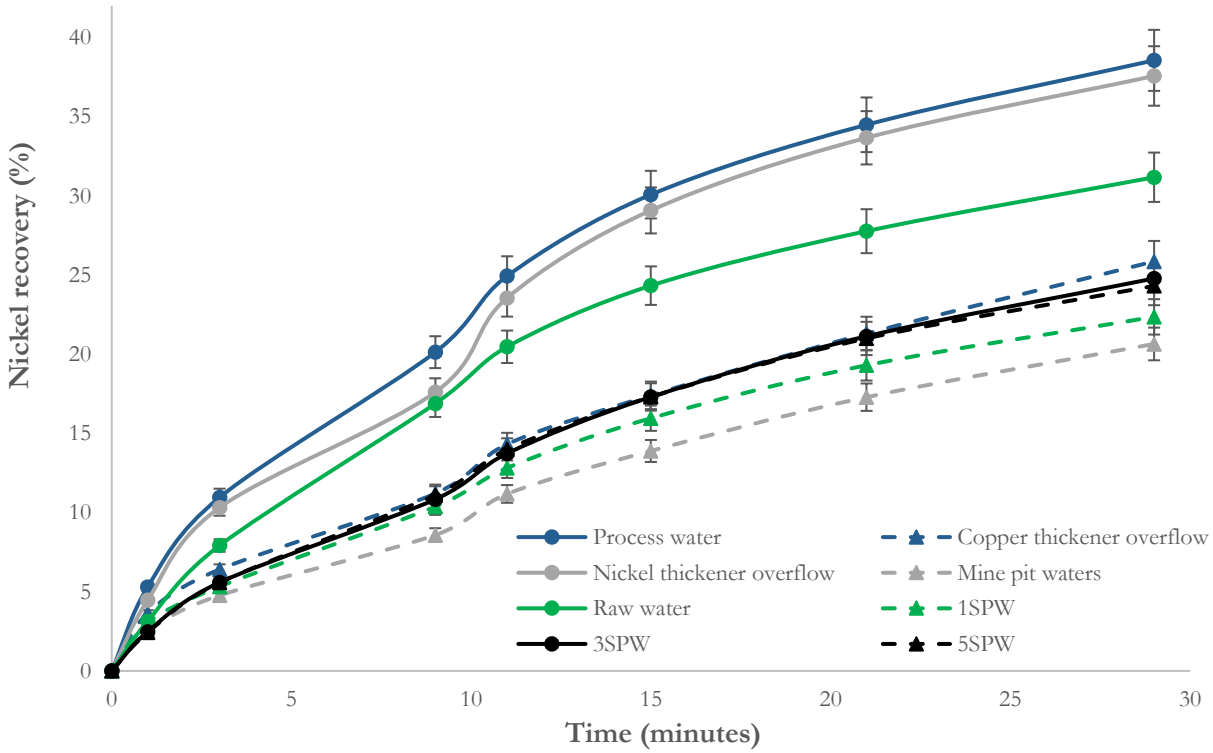

Figure 5.61: Nickel recovery vs. time for Ore B; no added collector and all tested waters

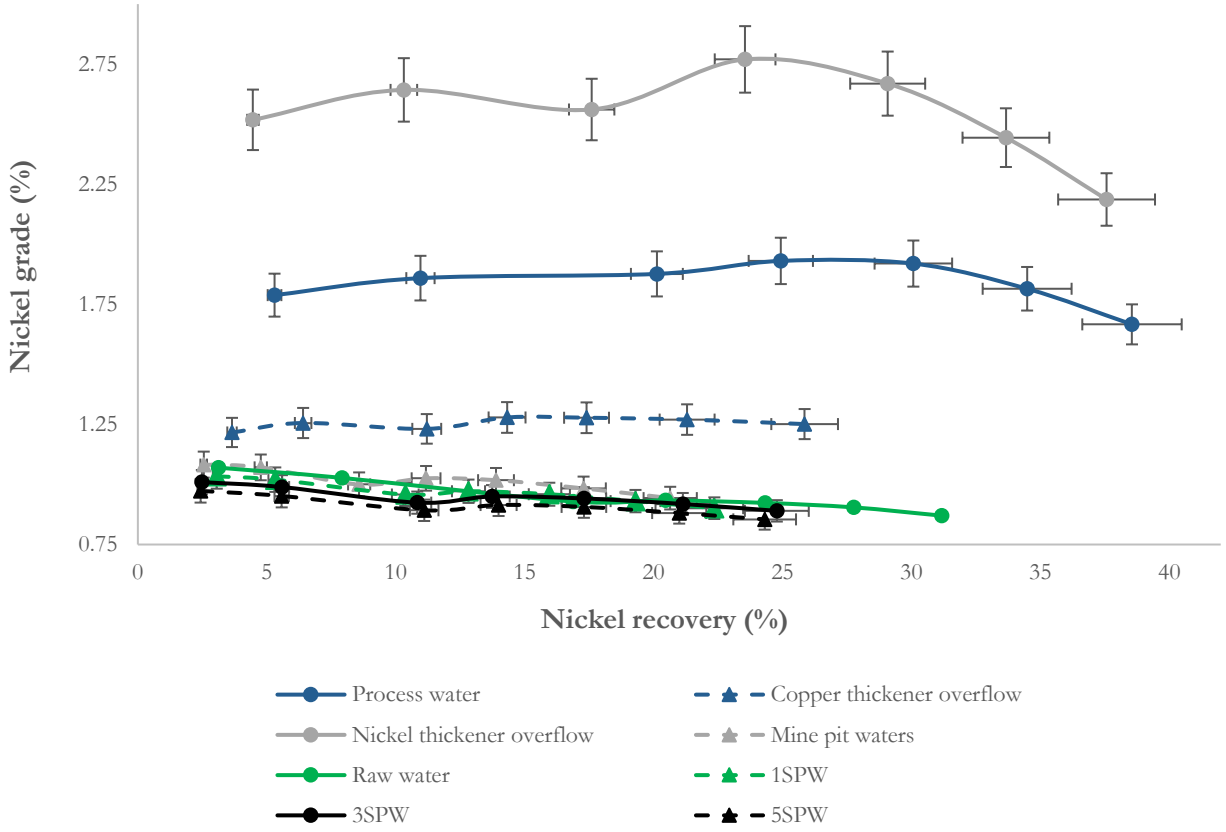

Figure 5.62: Nickel grade vs. recovery for Ore B; no added collector and all tested waters

Without added collector, the recovery of copper reached 81.9% when floated in process water; while for the worst-performing water in these conditions (the nickel thickener overflow) the overall recovery was 70.3%. The overall recovery of nickel was 38.5% in process water, and the worst performing water in this case was mine pit water, which reported an overall recovery of 20.6%.

For both copper and nickel, the flotation responses simulated in 3SPW and 5SPW mimicked the responses simulated in process water more closely than 1SPW. However, the responses for both these waters were even closer to mine pit waters (for the copper response) and the copper thickener overflow (for the nickel response). Copper grades were more favoured by process water, followed by the copper thickener overflow. On the other hand, nickel grades were more favoured by the nickel thickener overflow over other water types.

The final copper recoveries and grades for each water type are summarised by Figure 5.63, and the final nickel recoveries and grades are summarised by Figure 5.64.

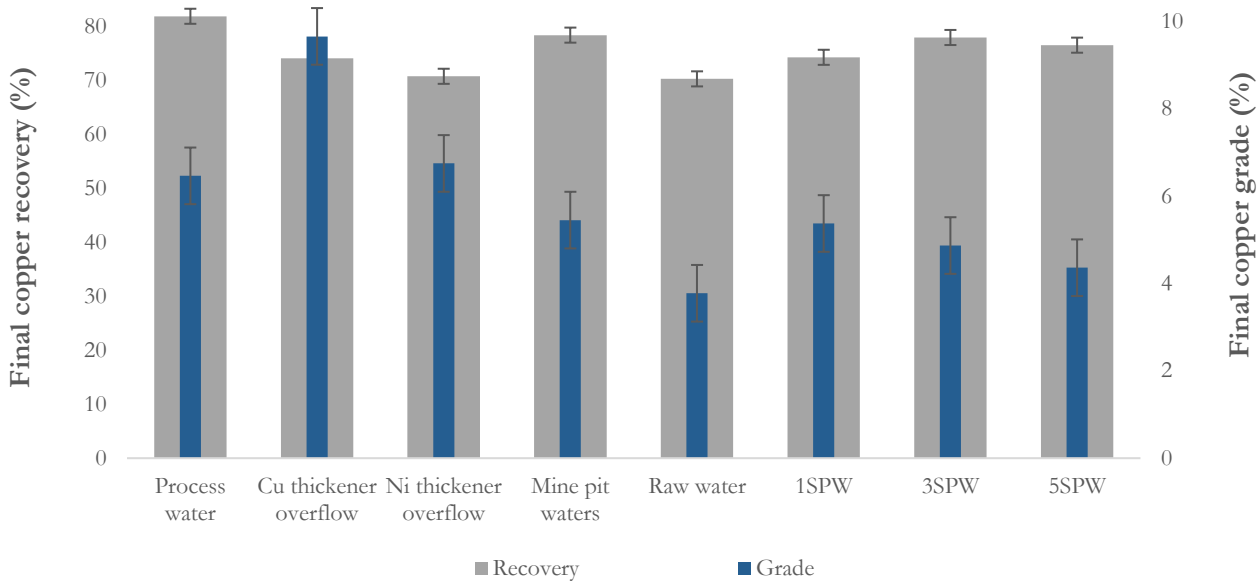

Figure 5.63: Final copper grade vs. recovery for Ore B; no added collector and all tested waters

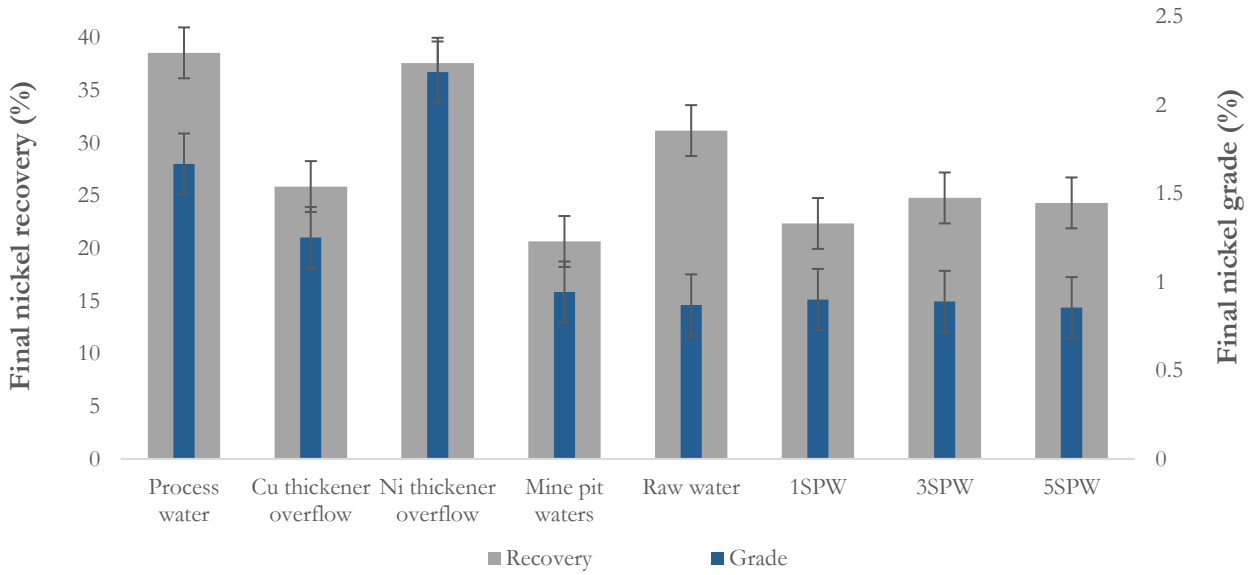

Figure 5.64: Final nickel grade vs. recovery for Ore B; no added collector and all tested waters

5.3.2. The Flotation Response of Ore B in Non-Recycled Water (Run 1)

For a dosage of 5 g/t aerophine and 100 g/t SIPX, Figure 5.65 illustrates the effect of water quality on the recovery of water to the concentrate, and Figure 5.66 illustrates effect of water quality on the recovery of solids. Additionally, Figure 5.67 depicts the effect of water quality on the recovery of total water, total solids and gangue materials to the concentrate, and Figure 5.68 shows the recovery of solids as a function of water recovery.

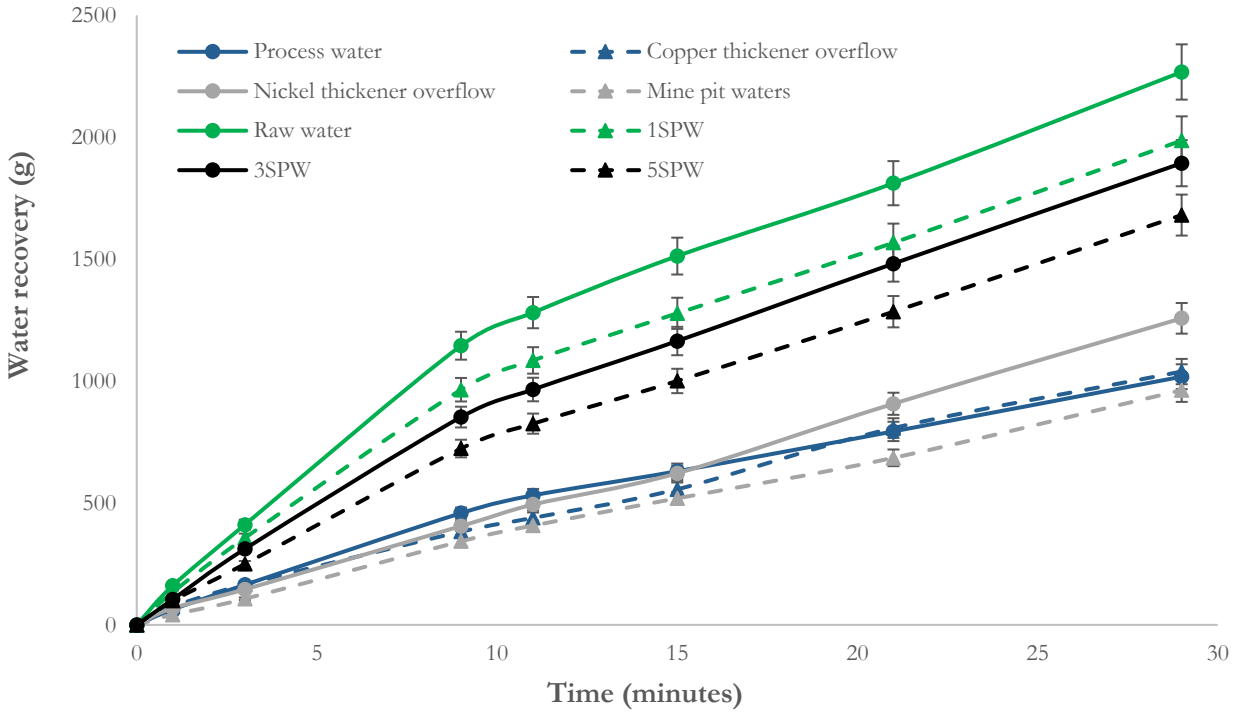

Figure 5.65: Water recovery vs. time for Ore B; 5 g/t aerophine, 100 g/t SIPX and all tested waters

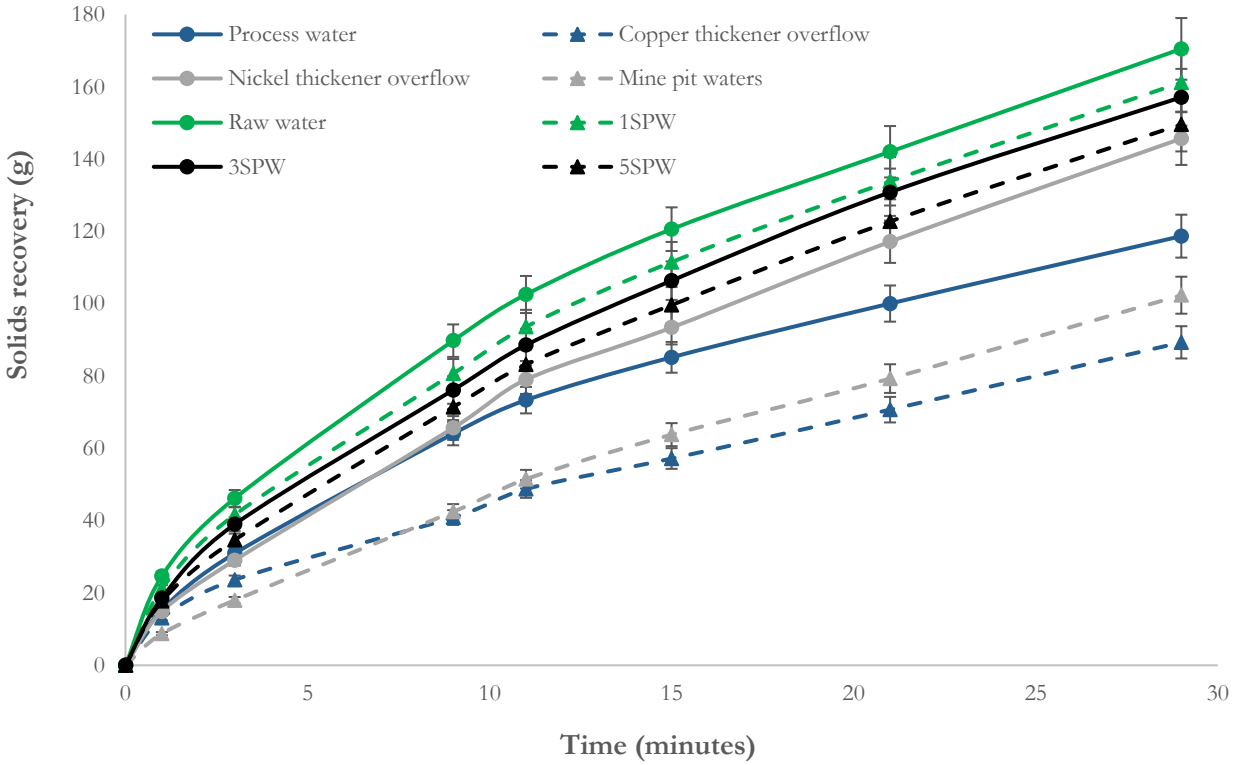

Figure 5.66: Solids recovery vs. time for Ore B; 5 g/t aerophine, 100 g/t SIPX and all tested waters

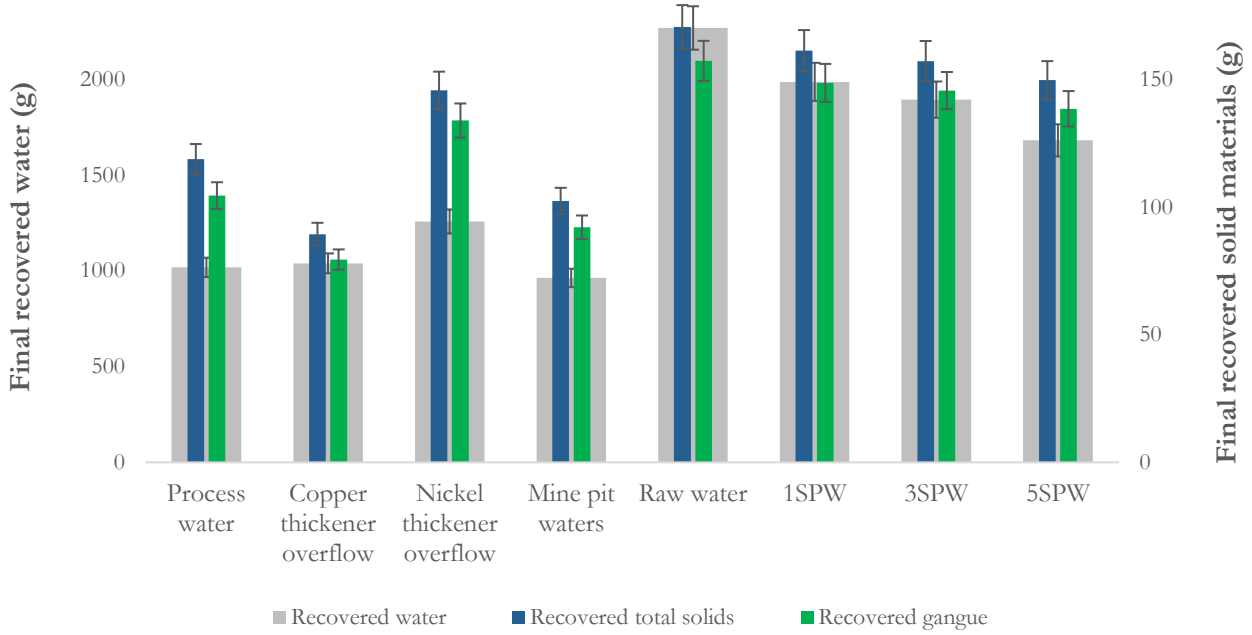

Figure 5.67: Total solids and gangue material vs. water recovery for Ore B; 5 g/t, 100 g/t SIPX and all tested waters

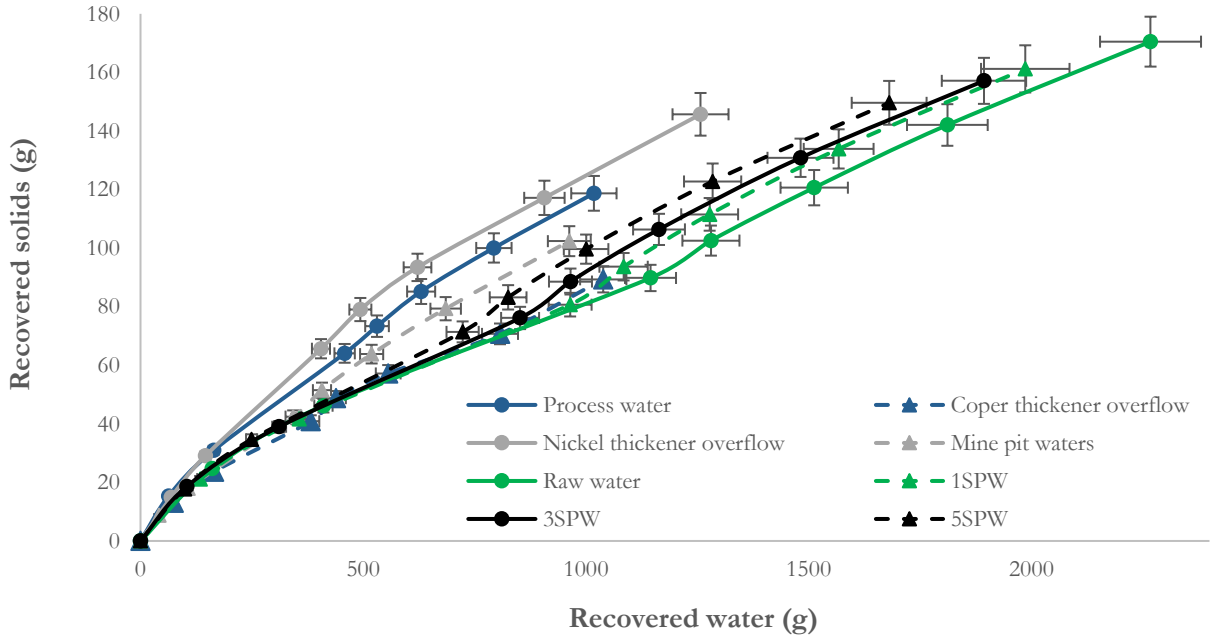

Figure 5.68: Solids vs. water recovery for Ore B; 5 g/t aerophine, 100 g/t SIPX and all tested waters

As was observed with the collectorless trends, increased water recoveries were complemented by increased gangue mineral recoveries, and flotation in raw water resulted in the highest amount of water and solids recovered to the concentrate. However, the total water and solid recoveries were lower than those determined in the collectorless runs; this, with the exception of the nickel thickener overflow

trend, which showed higher water and gangue mineral recoveries, and 1SPW, which only showed higher gangue mineral recoveries. The biggest reduction in the amount of recovered water corresponds with the biggest reduction in the amount of gangue recovered; this, for the mine pit waters, which exhibited a reduction factor of 2.56 for water and 1.35 for gangue materials. As a result, and overall, the grades were improved when compared to the collectorless tests.

For 5 g/t aerophine and 100 g/t SIPX in all tested waters, Figure 5.69 and Figure 5.71 show the respective recovery of copper, and the recovery of nickel to the concentrate as a function of time. Figure 5.70 and Figure 5.72 illustrate the grade vs. recovery curves for copper and nickel, respectively. Both copper and nickel grades were most favoured by process water. Raw water outperformed the other waters in terms of the recovery responses of both copper and nickel; the recovery of copper reached 73.8% in the first 9 minutes, with the only collector being aerophine. After SIPX was added, the overall recovery was 85.4%. The overall nickel recovery reached was 46.2%. When compared to the other synthetic plant waters, the responses in 5SPW more closely resembled that in process water.

The recovery of copper when floated in the nickel thickener overflow increased by 18.3%, from 56.7% at 9 minutes of flotation to 75.0% at 29 minutes. Moreover, the recovery of nickel increased by 19.6%, from 26.6% at 9 minutes of flotation to 46.2% at 29 minutes. In this regard, the nickel thickener overflow proved the most improved by the addition of SIPX.

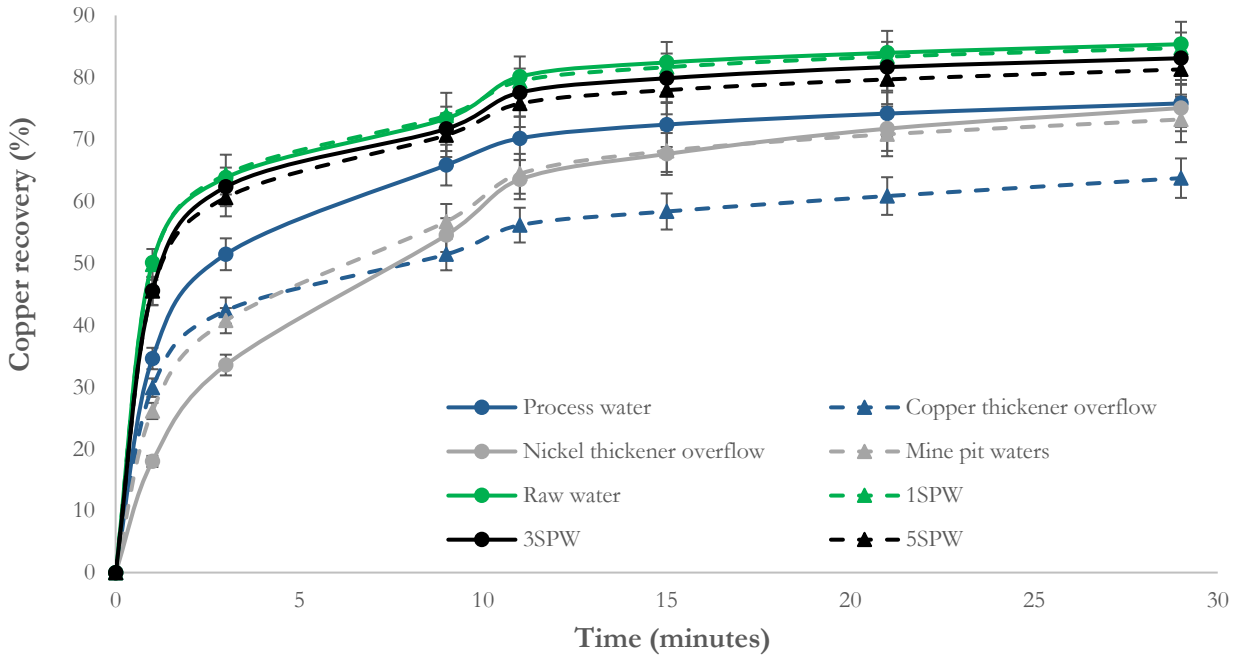

Figure 5.69: Copper recovery vs. time for Ore B; 5 g/t aerophine, 100 g/t SIPX and all tested waters

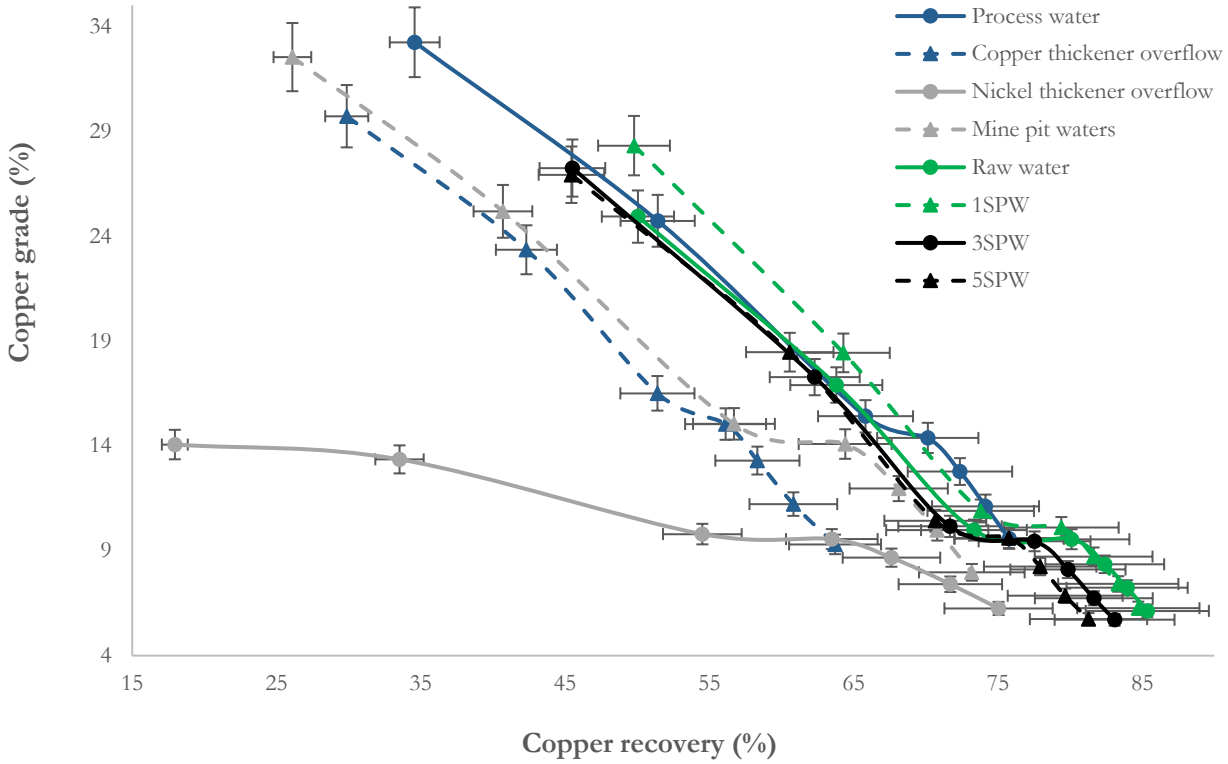

Figure 5.70: Copper grade vs. recovery for Ore B; 5 g/t aerophine, 100 g/t SIPX and all tested waters

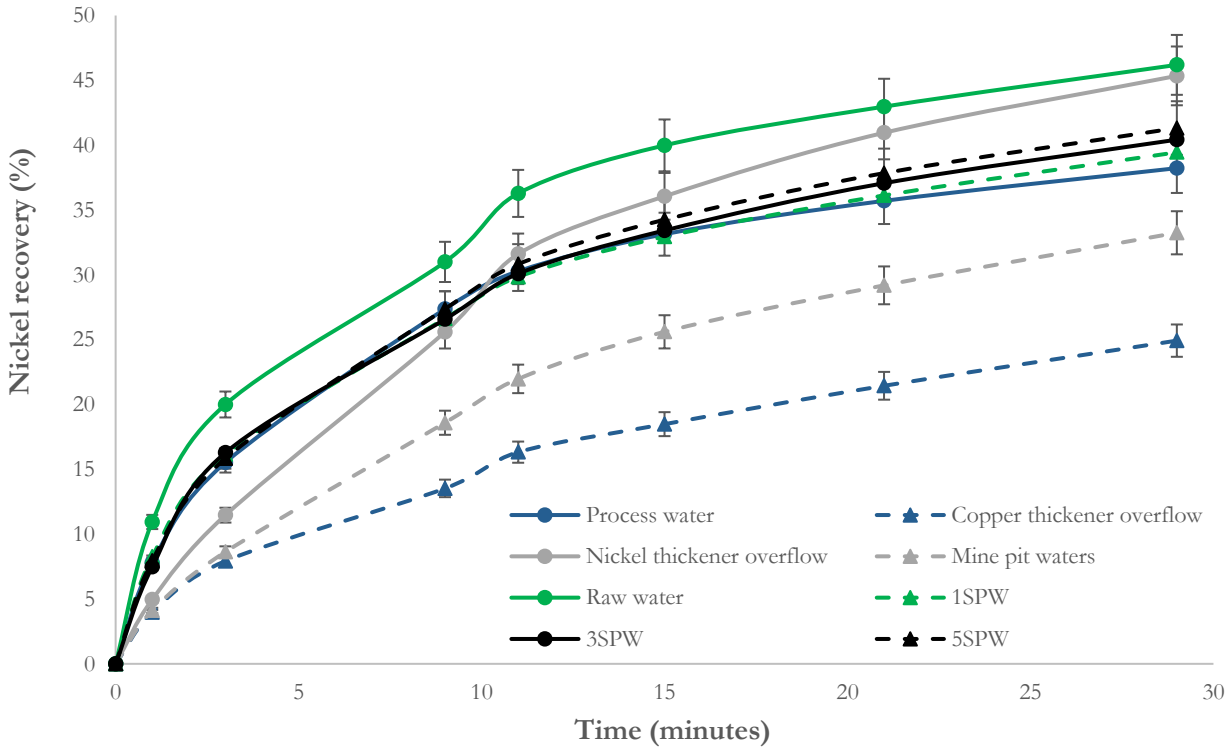

Figure 5.71: Nickel recovery vs. time for Ore B; 5 g/t aerophine, 100 g/t SIPX and all tested waters

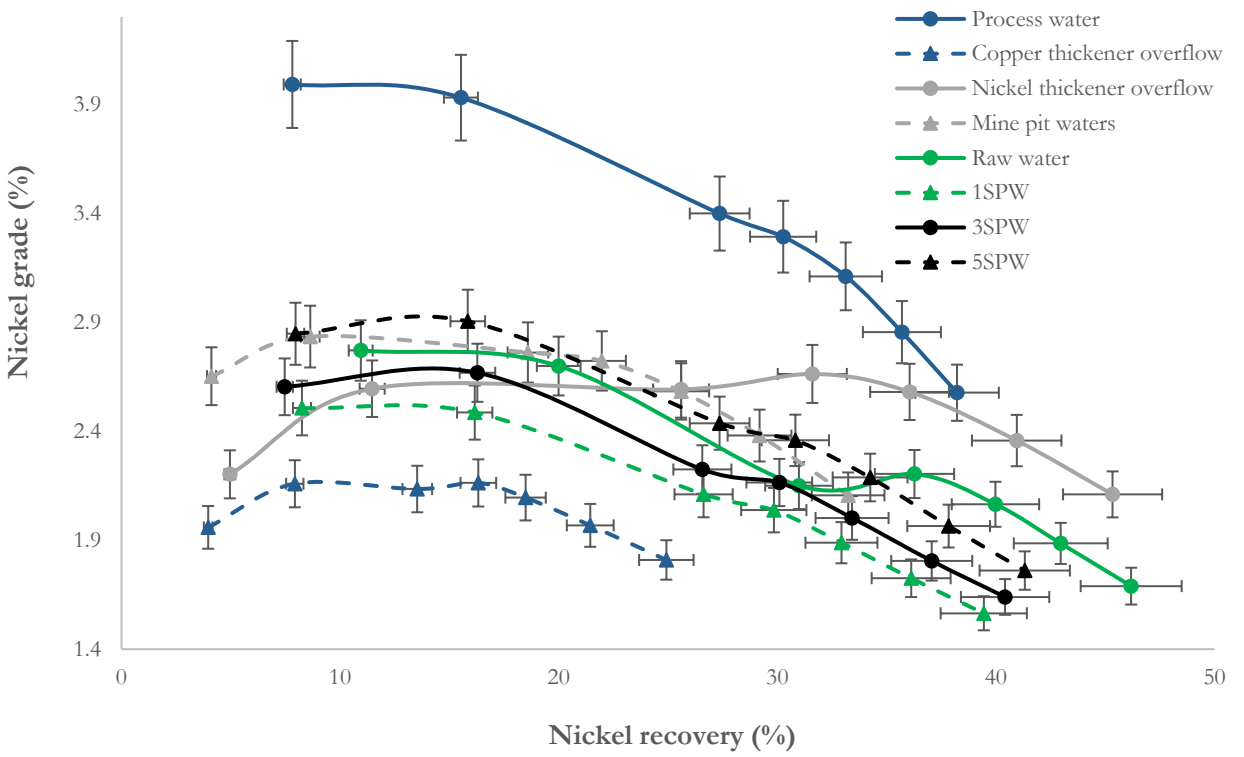

Figure 5.72: Nickel grade vs. recovery for Ore B; 5 g/t aerophine, 100 g/t SIPX and all tested waters

The final copper recoveries and grades for each water type are summarised by Figure 5.73, and the final nickel recoveries and grades are summarised by Figure 5.74.

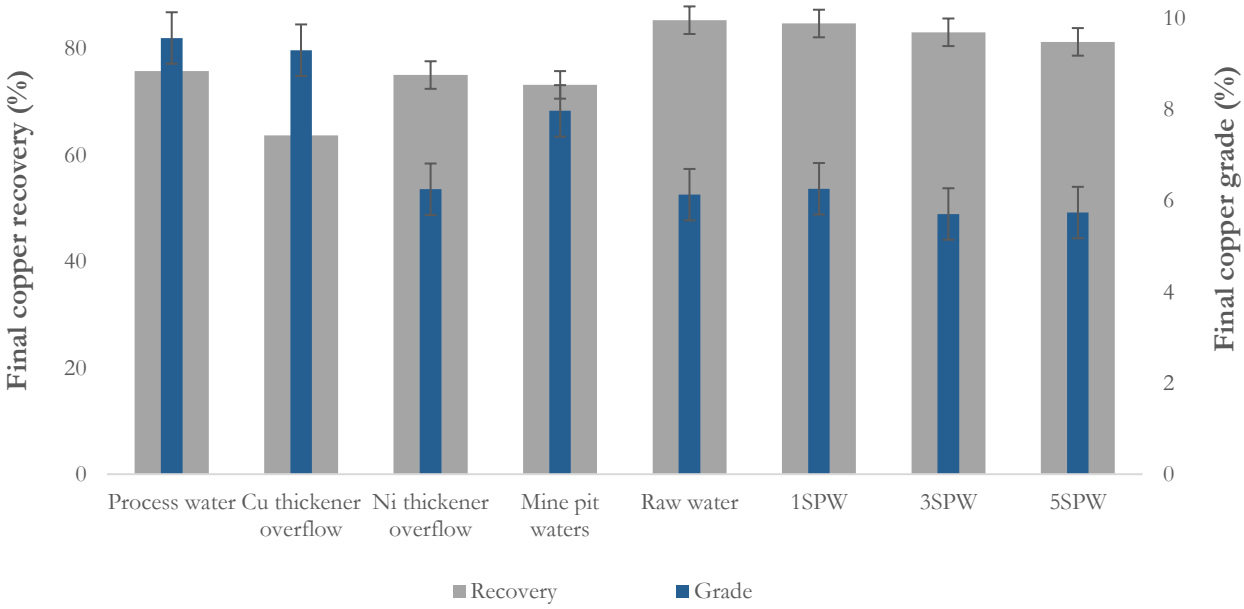

Figure 5.73: Final copper grade vs. recovery for Ore B; 5 g/t aerophine, 100 g/t SIPX and all tested waters

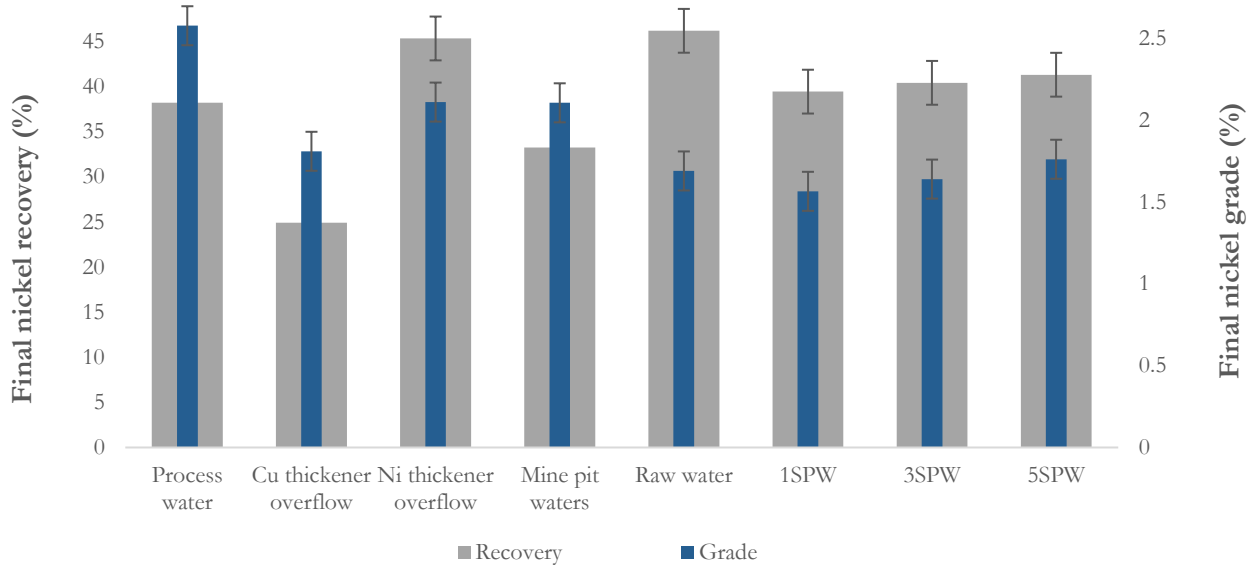

Figure 5.74: Final nickel grade vs. recovery for Ore B; 5 g/t aerophine, 100 g/t SIPX and all tested waters

The adsorption tests show that the highest amount of SIPX was adsorbed in raw water and the nickel thickener overflow; while at the same time, the least amount was adsorbed in copper thickener overflow, as illustrated in Figure 5.75.

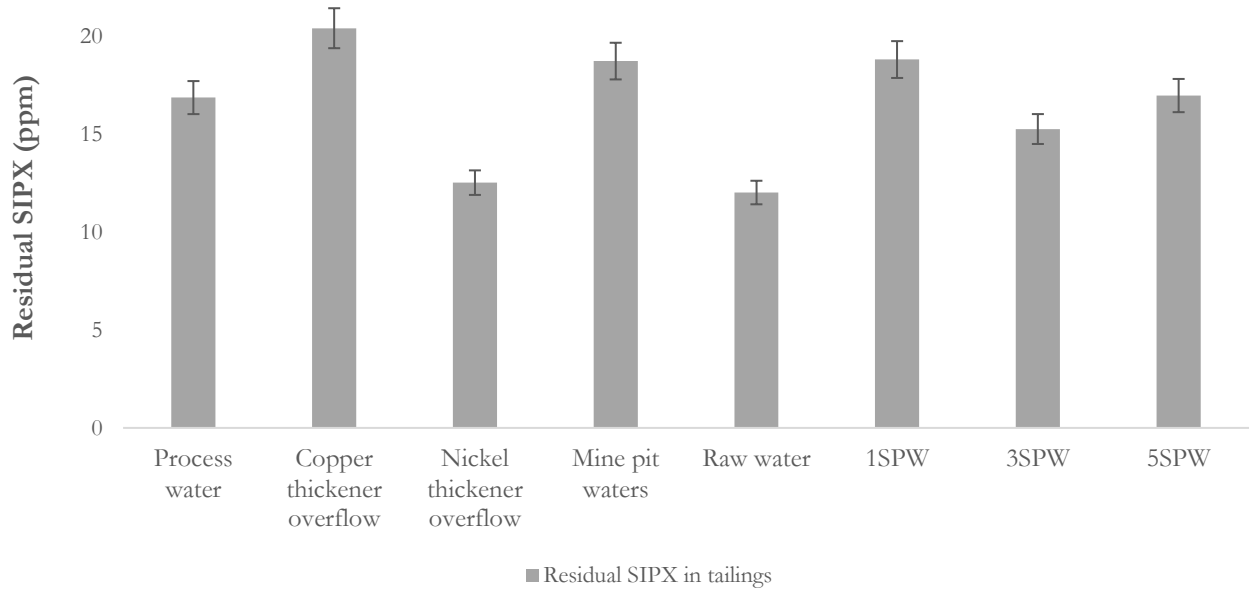

Figure 5.75: The SIPX concentration remaining in the tailings, for an initial dosage of 100 g/t SIPX in Run 1

For the synthetic plant waters, the overall adsorption trend can be summarised by Figure 5.76, which depicts the amount of residual xanthate in the tailings when compared to the fraction of the initial concentration that was adsorbed in each tested case. Fitting a linear relationship to the adsorption

amounts across the different waters proved ineffective—as is indicated by the R-squared value of 0.269.

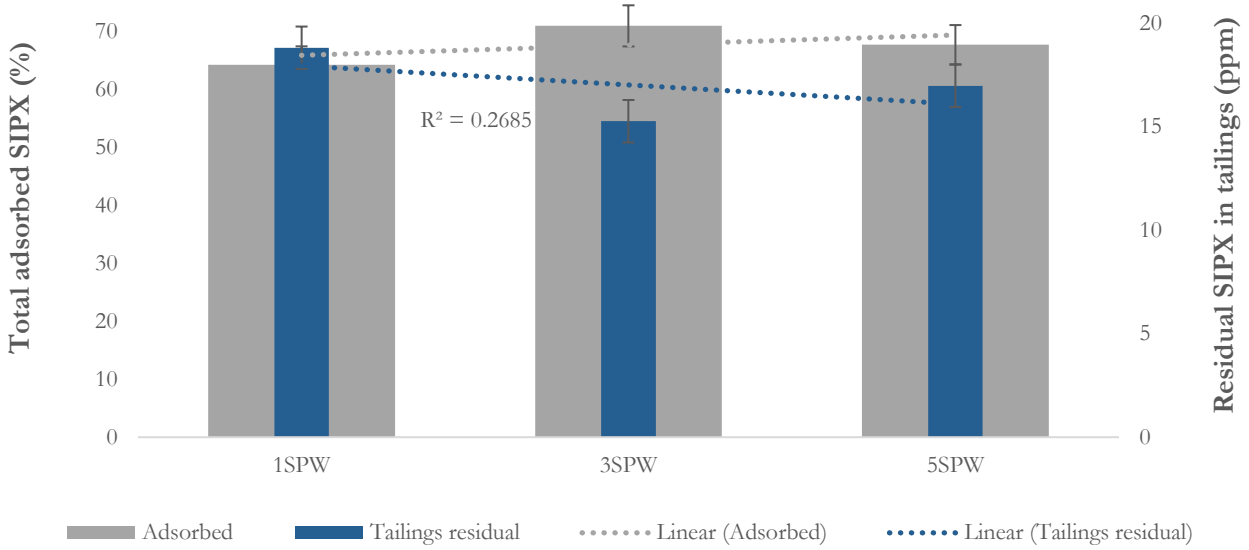

Figure 5.76: A summary of the amount of adsorbed SIPX in mg/litres (=ppm), plotted against the fraction of the initial dosage that was adsorbed in the synthetic plant waters in Run 1

Figure 5.77 plots the amount of xanthate that was adsorbed against the final water recovery in all tested waters, and Figure 5.78 plots the total adsorbed xanthate against the total amount of recovered solids and gangue minerals.

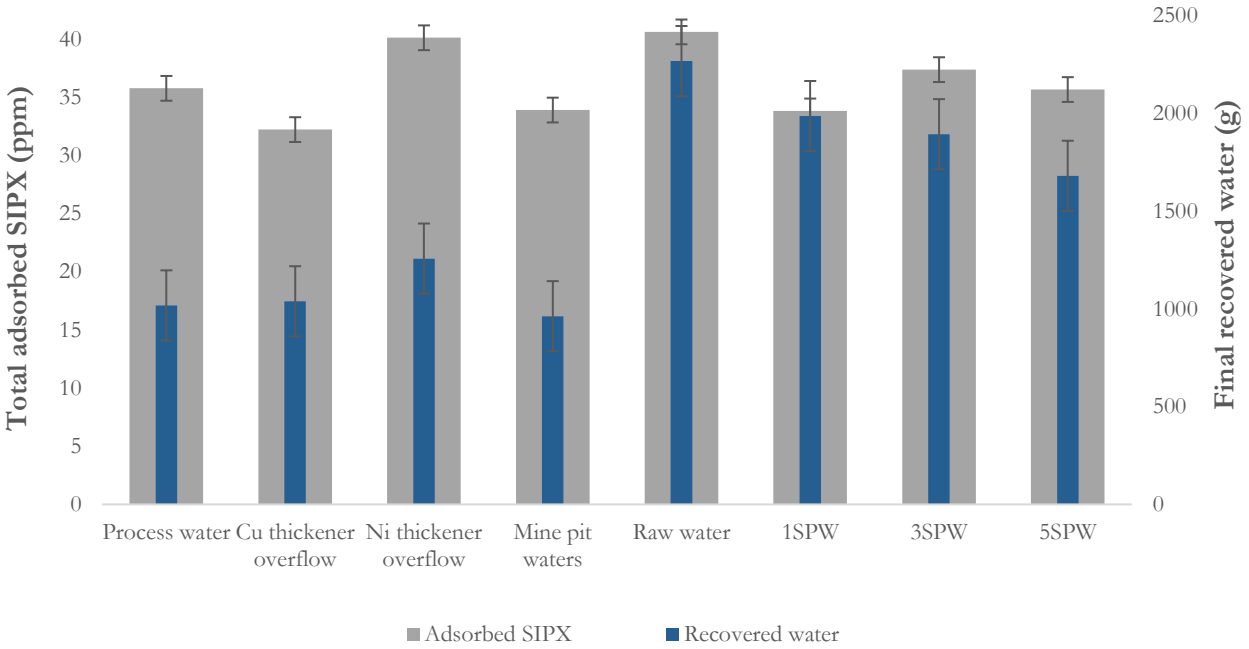

Figure 5.77: Total adsorbed SIPX plotted against final water recovery for Ore B in all tested waters in Run 1

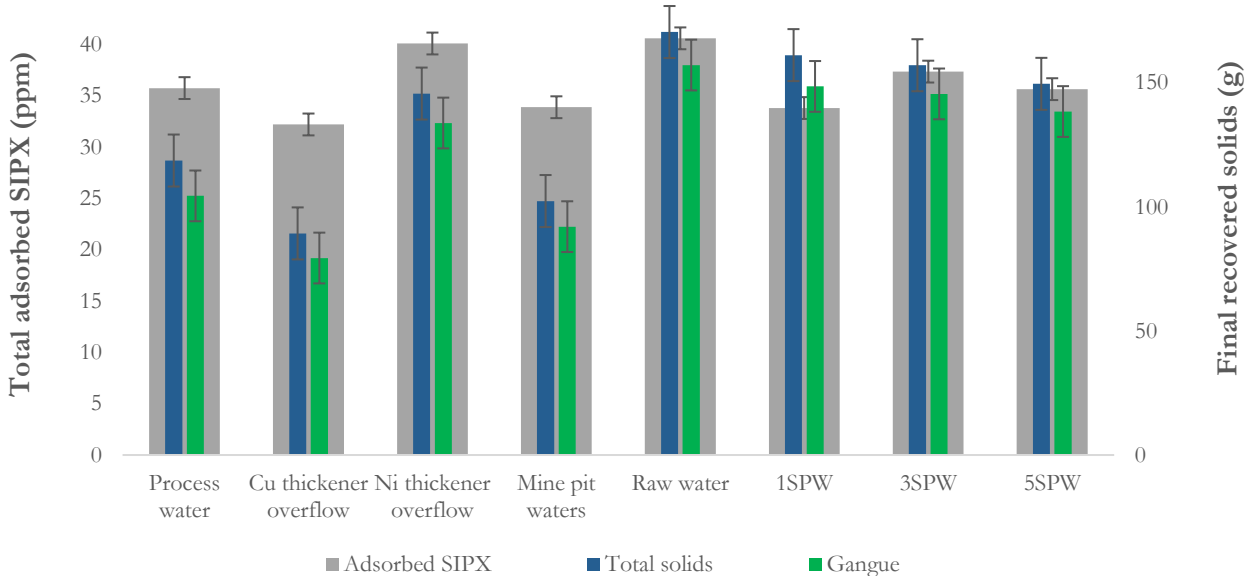

Figure 5.78: Total adsorbed SIPX plotted against final solids recovery for Ore B in all tested waters in Run 1

The most amount of collector was adsorbed in raw water and at the same time, the most amount of water and solids were recovered in raw water. On the other hand, the least amount of collector was adsorbed in the copper thickener overflow, but the least amount of water was recovered in the mine pit waters, while the lowest mass of solids was recovered in the copper thickener overflow.

Figure 5.79 shows the final copper and nickel recoveries against the total amount of adsorbed SIPX. The figure suggests that for the synthetic plant waters, the degree of collector adsorption did not significantly affect the final copper and nickel recoveries. But for the actual plant waters, higher adsorption coincided with higher copper and nickel recoveries.

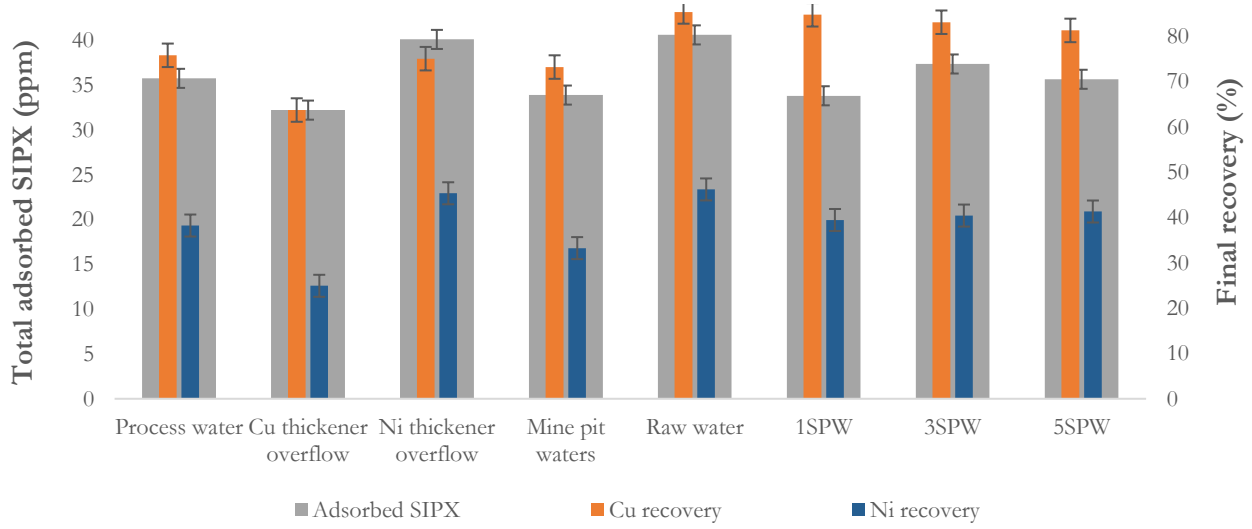

Figure 5.79: Total adsorbed SIPX plotted against final Cu and Ni recoveries for all tested waters in Run 1

Figure 5.80 shows the final copper and nickel grades against the total amount of adsorbed xanthate. Among the synthetic plant waters, more SIPX was adsorbed in 3SPW, followed by 5SPW; at the same time, the highest copper grade was achieved in 1SPW, followed by 3SPW, but the highest nickel grades were achieved in 3SPW and 5SPW. For the actual plant waters, the highest adsorption amounts did not coincide the highest grade for either metal.

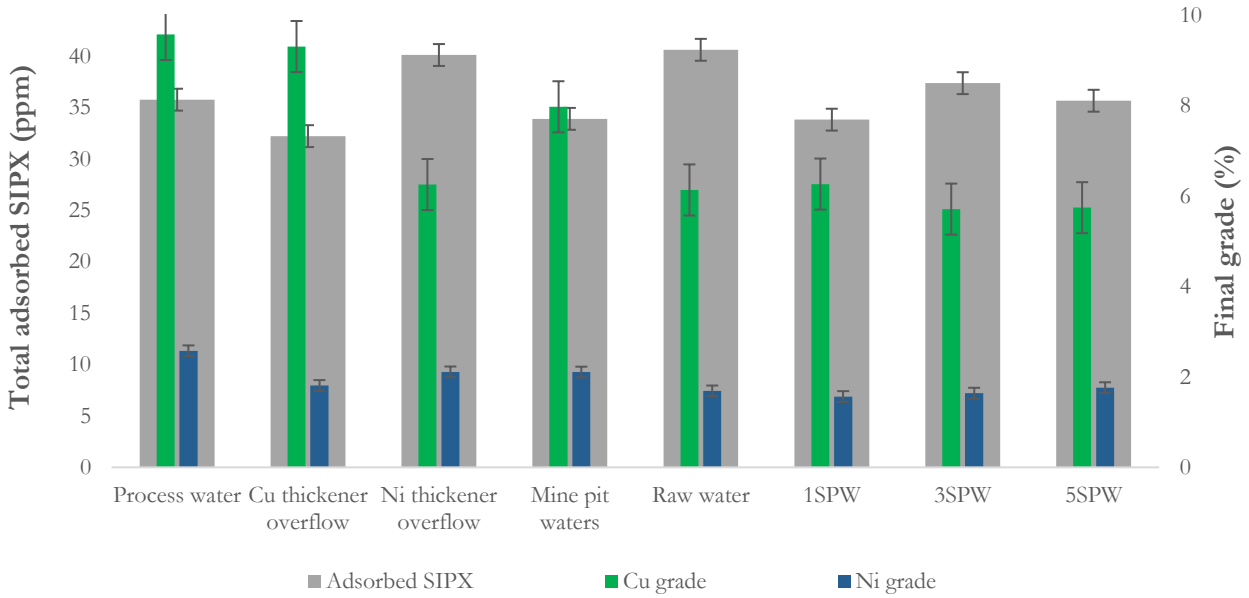

Figure 5.80: Total adsorbed SIPX plotted against final Cu and Ni grades for all tested waters in Run 1

5.3.3. The Flotation Response of Ore B in Once-Recycled Water (Run 2)

As a result of different concentrations of SIPX being adsorbed in the first run, different concentrations of SIPX were therefore present at the start of the recycle runs, depending on the water. Table 5.5 shows the concentrations of SIPX at the beginning of flotation for the tested waters.

Table 5.5: SIPX concentrations in the feed for the recycle runs

Water type	Process	Cu thickener	Ni thickener	Mine pit	Raw	1SPW	3SPW	5SPW
SIPX (ppm)	11.9	16.9	9.61	12.6	10.5	14.1	11.4	12.6

For a Run 2 dosage of 5 g/t aerophine and 100 g/t SIPX, Figure 5.81 illustrates the effect of water quality on the recovery of water to the concentrate, and Figure 5.82 illustrates effect of water quality on the recovery solids. Additionally, Figure 5.83 depicts the effect of water quality on the recovery of total water, total solids and gangue materials to the concentrate, and Figure 5.84 shows the recovery of solids as a function of water recovery.

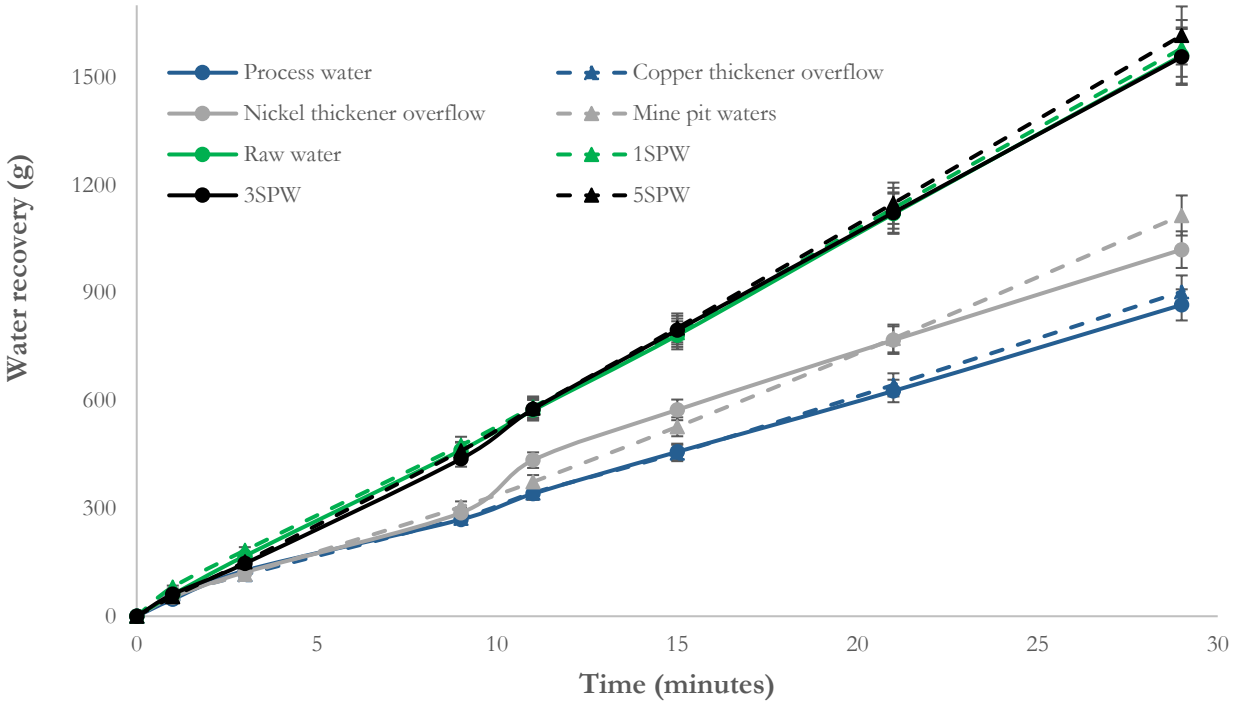

Figure 5.81: Water recovery vs. time for Ore B; 5 g/t aerophine, 100 g/t SIPX and all once-recycled waters

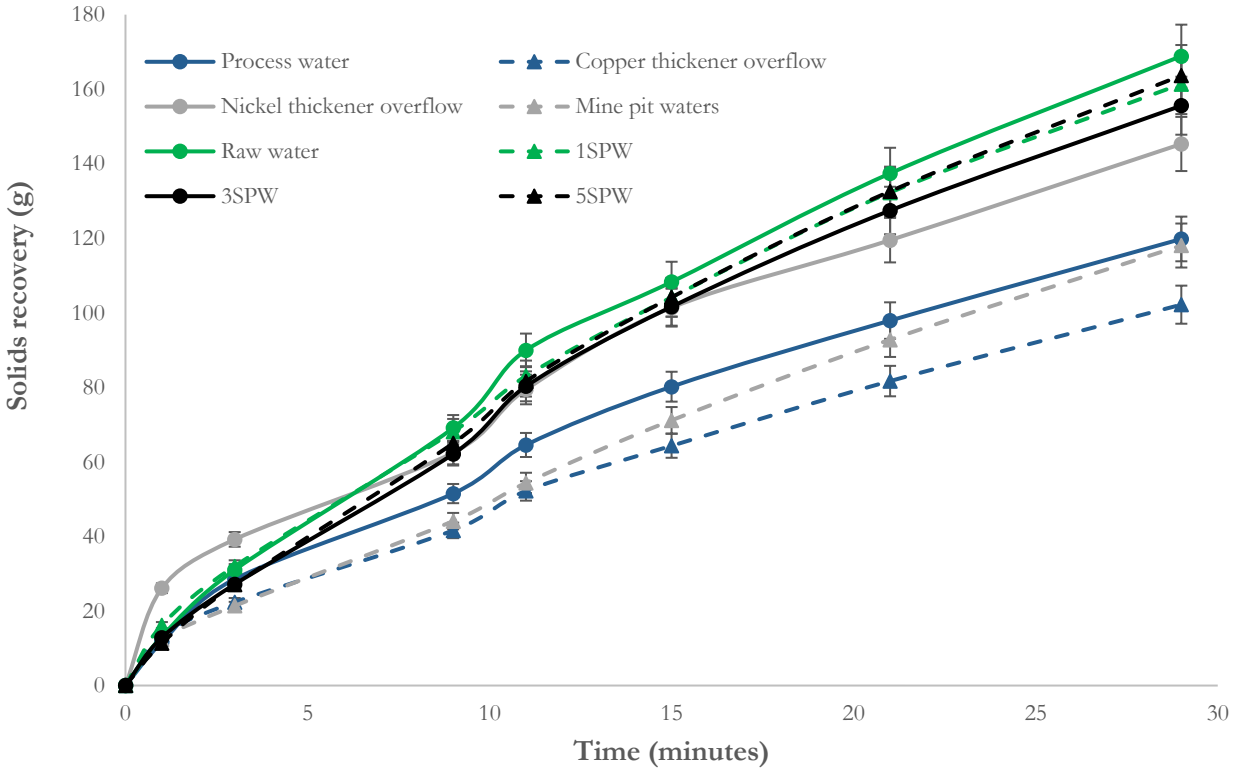

Figure 5.82: Solids recovery vs. time for Ore B; 5 g/t aerophine, 100 g/t SIPX and all once-recycled waters

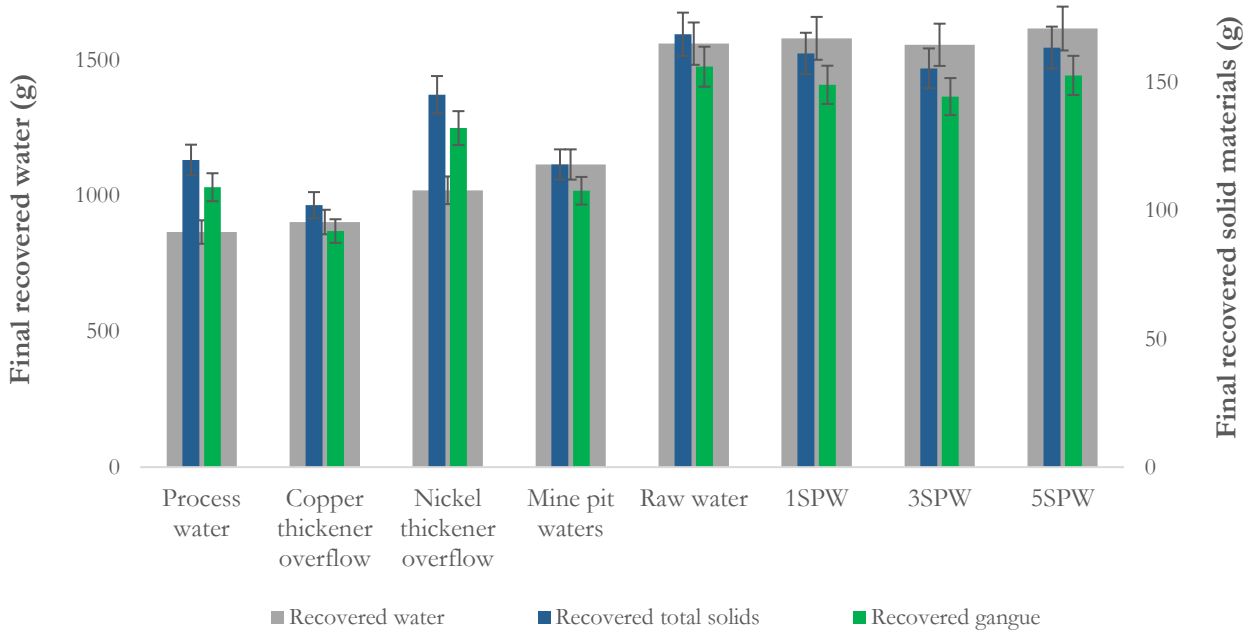

Figure 5.83: Total solids and gangue material vs. water recovery for Ore B; 5 g/t, 100 g/t SIPX and all once-recycled waters

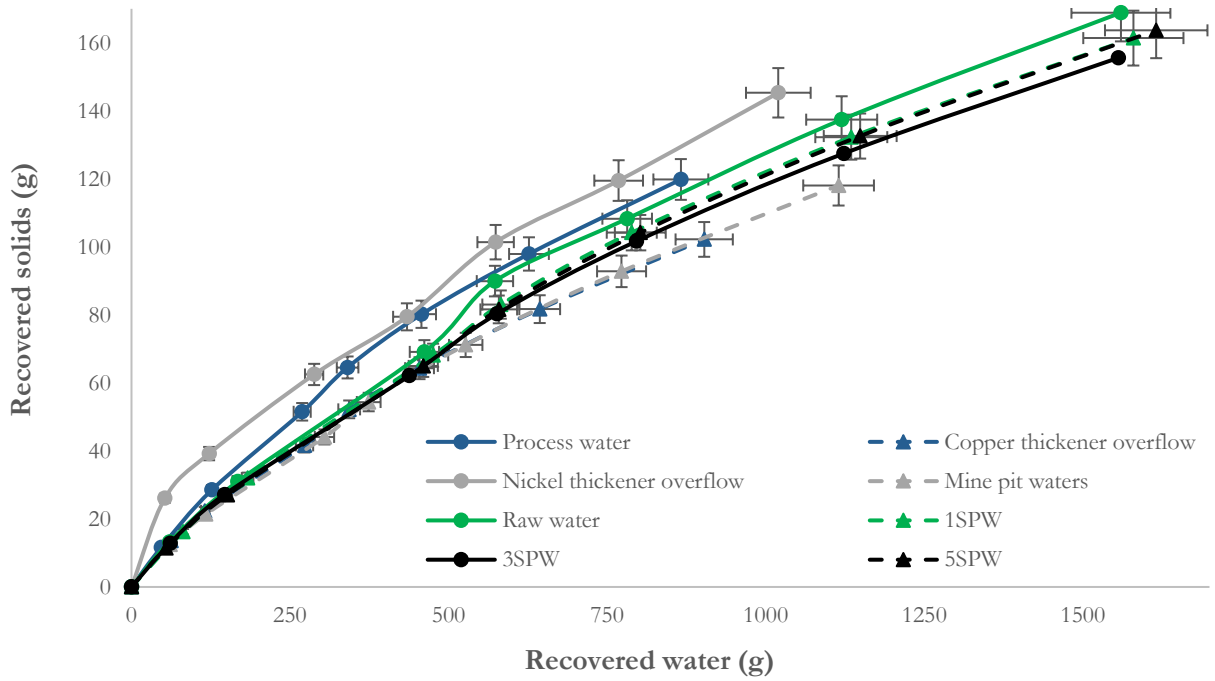

Figure 5.84: Solids vs. water recovery for Ore B; 5 g/t aerophine, 100 g/t SIPX and all once-recycled waters

When compared to their non-recycle counterparts, the recycle tests exhibited lower water recoveries; however, the gangue mineral recoveries were increased, suggesting reduced selectivity for all waters except the nickel thickener overflow, which had a gangue mineral recovery reduction of 0.827%.

Figure 5.85 and Figure 5.87 show the respective recovery of copper, and the recovery of nickel to the concentrate as a function of time. Figure 5.86 and Figure 5.88 illustrate the grade vs. recovery curves for copper and nickel, respectively.

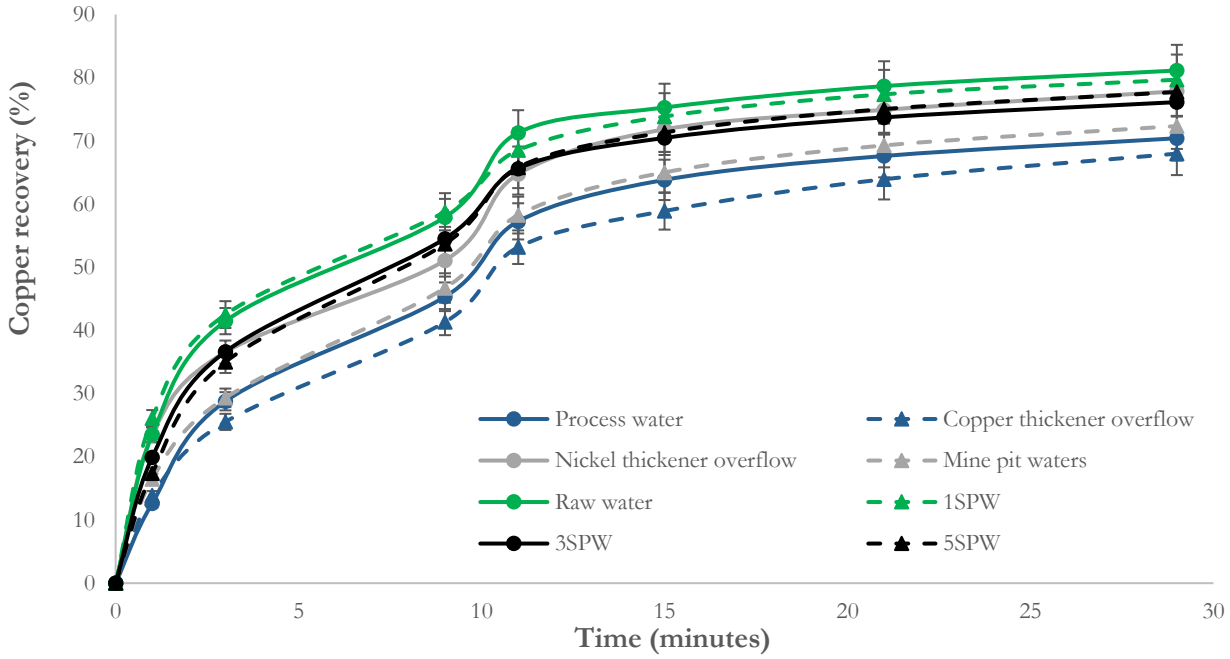

Figure 5.85: Copper recovery vs. time for Ore B; 5 g/t aerophine, 100 g/t SIPX and all once-recycled waters

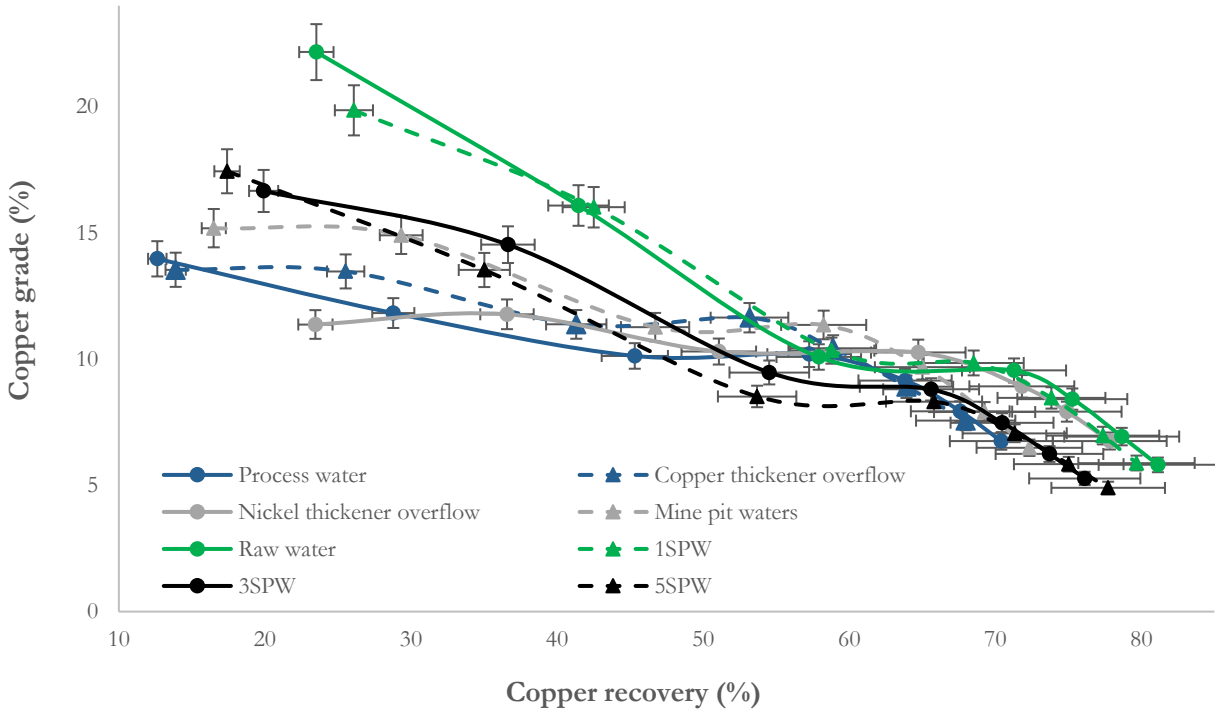

Figure 5.86: Copper grade vs. recovery for Ore B; 5 g/t aerophine, 100 g/t SIPX and all once-recycled waters

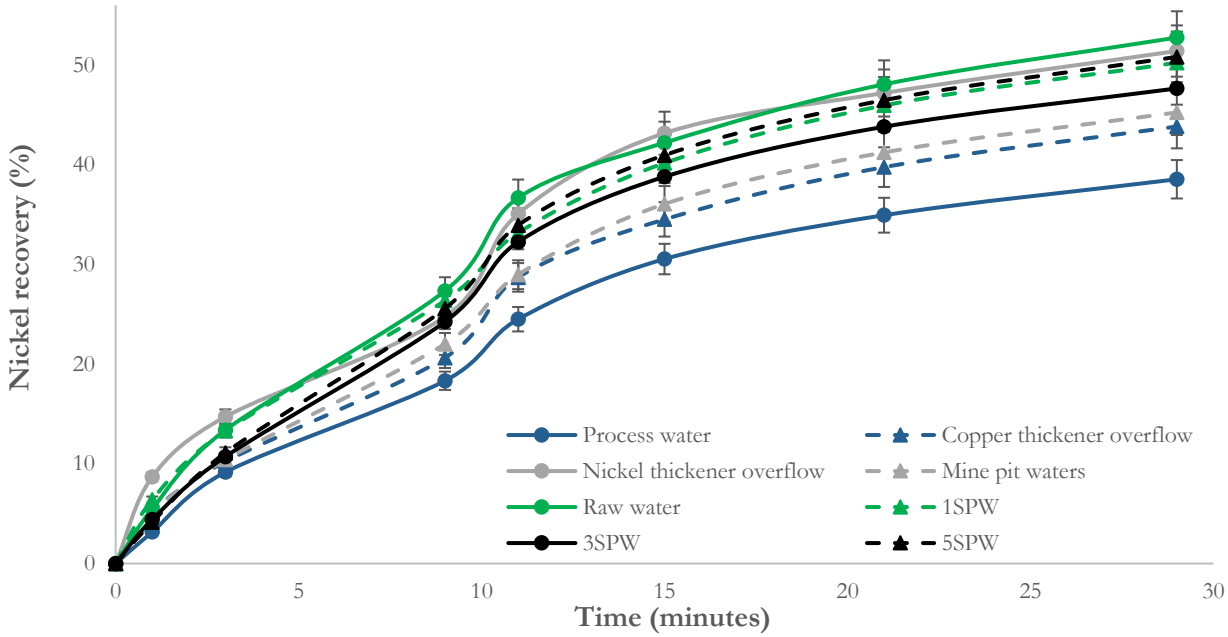

Figure 5.87: Nickel recovery vs. time for Ore B; 5 g/t aerophine, 100 g/t SIPX and all once-recycled waters

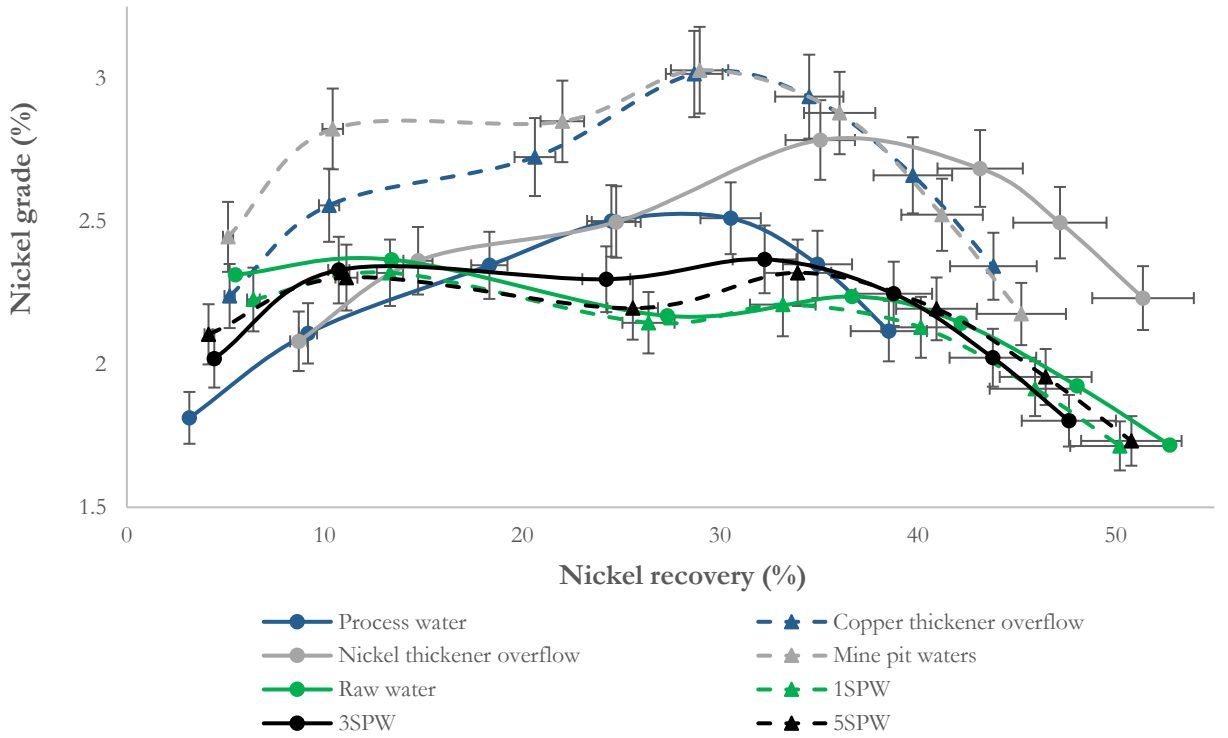

Figure 5.88: Nickel grade vs. recovery for Ore B; 5 g/t aerophine, 100 g/t SIPX and all once-recycled waters

Raw water outperformed the other waters in terms of the recovery responses of both metals; the recovery of copper reached 58.8% in the first 9 minutes of flotation, with the initial xanthate

concentration being 10.5 ppm. After more xanthate was added, the overall recovery was 81.1%. In the first 9 minutes, the nickel recovery was 27.3%, and at the end it was 52.7%. When compared to the other synthetic plant waters, the responses in 3SPW more closely resembled those in process water. Of the three investigated systems, the recycle simulation yielded the highest overall nickel recovery and lowest overall copper recovery. The final copper recoveries and grades for each water type are summarised by Figure 5.89, and the final nickel recoveries and grades are summarised by Figure 5.90.

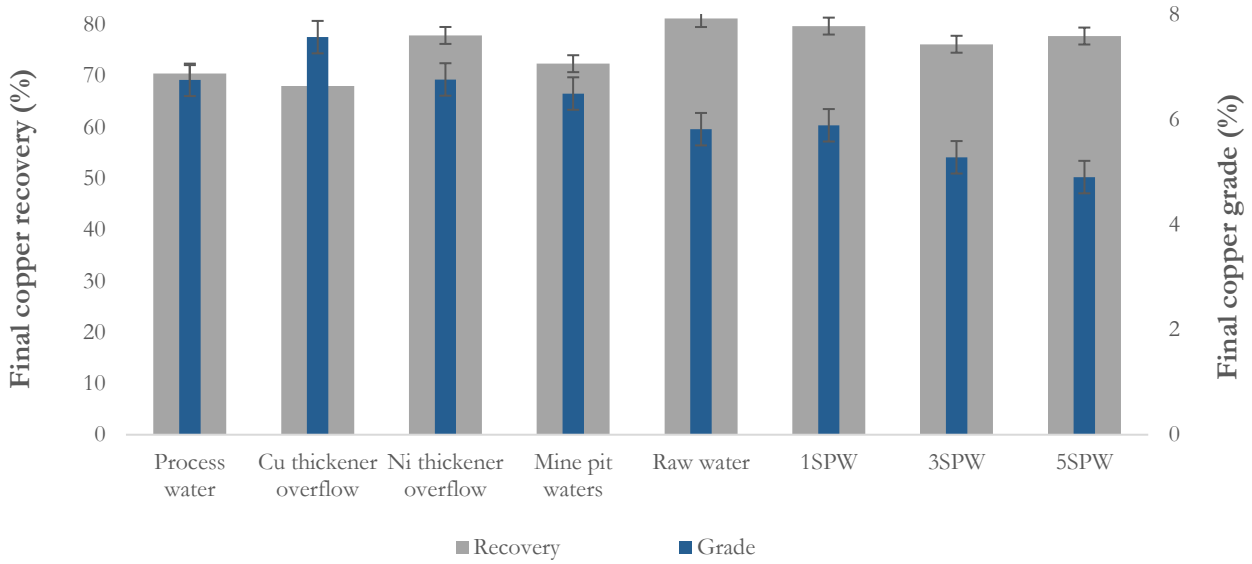

Figure 5.89: Final copper grade vs. recovery for Ore B; 5 g/t aerophine, 100 g/t SIPX and all once-recycled waters

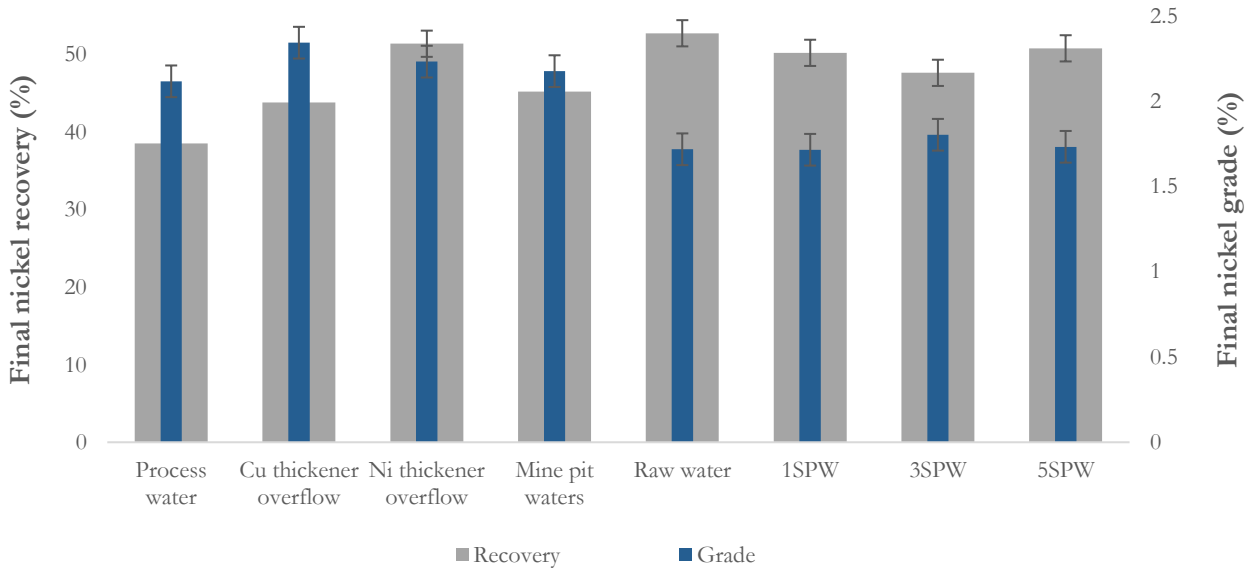

Figure 5.90: Final nickel grade vs. recovery for Ore B; 5 g/t aerophine, 100 g/t SIPX and all once-recycled waters

The adsorption tests (Figure 5.91) show that the most SIPX was adsorbed in nickel thickener overflow, followed by adsorption in raw water and process. Raw water and nickel thickener overflow flotation resulted in the highest nickel recoveries. However, the process water flotation resulted in the poorest nickel recovery, suggesting poor SIPX selectivity when the collector is in the presence of process water that contains residual xanthate. Despite the copper thickener overflow containing the most amount of SIPX at the start of flotation, at the end, the least amount of SIPX was adsorbed in the copper thickener overflow. This corresponds with poorer nickel recovery when compared to the other waters.

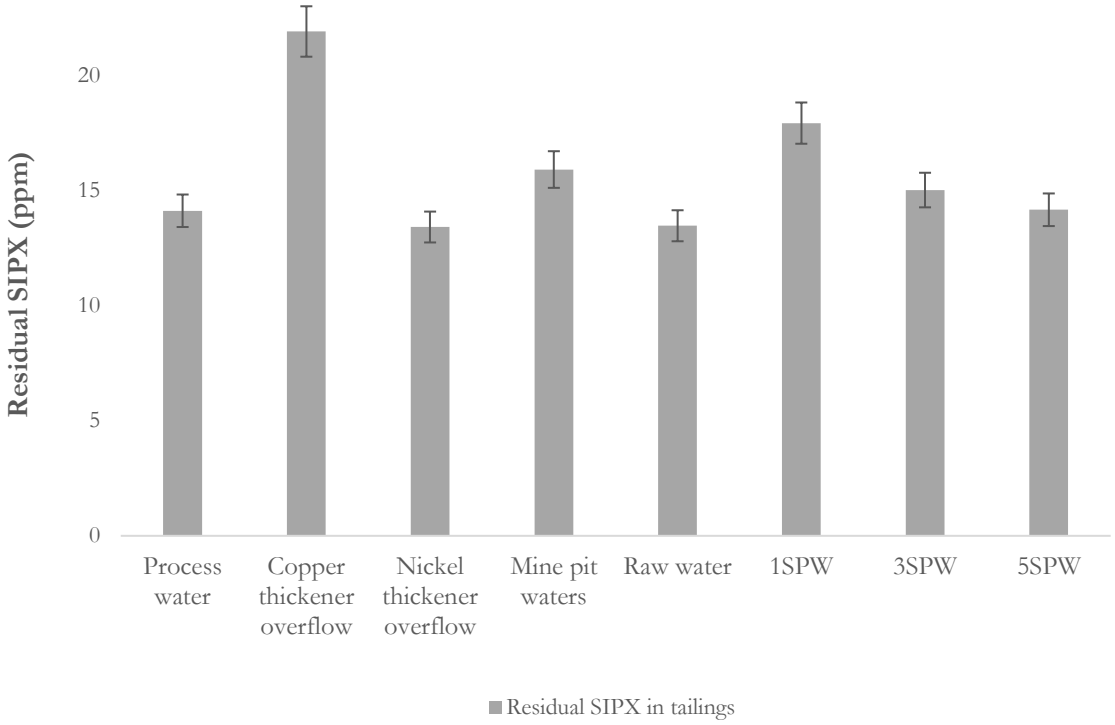

Figure 5.91: The SIPX concentration remaining in the tailings, for an initial dosage of 100 g/t SIPX in Run 2

For the synthetic plant waters, the overall adsorption trend can be summarised by Figure 5.92, which depicts the amount of residual xanthate in the tailings when compared to the fraction of the initial concentration that was adsorbed in each tested case.

Unlike Run 1, it is apparent that across the synthetic plant waters, the adsorbed amount of SIPX and adsorbed fraction can each be better approximated by a linear relationship. The shared R-squared value is closer to 1 (at 0.909). This indicates sufficient fits for the linear approximations.

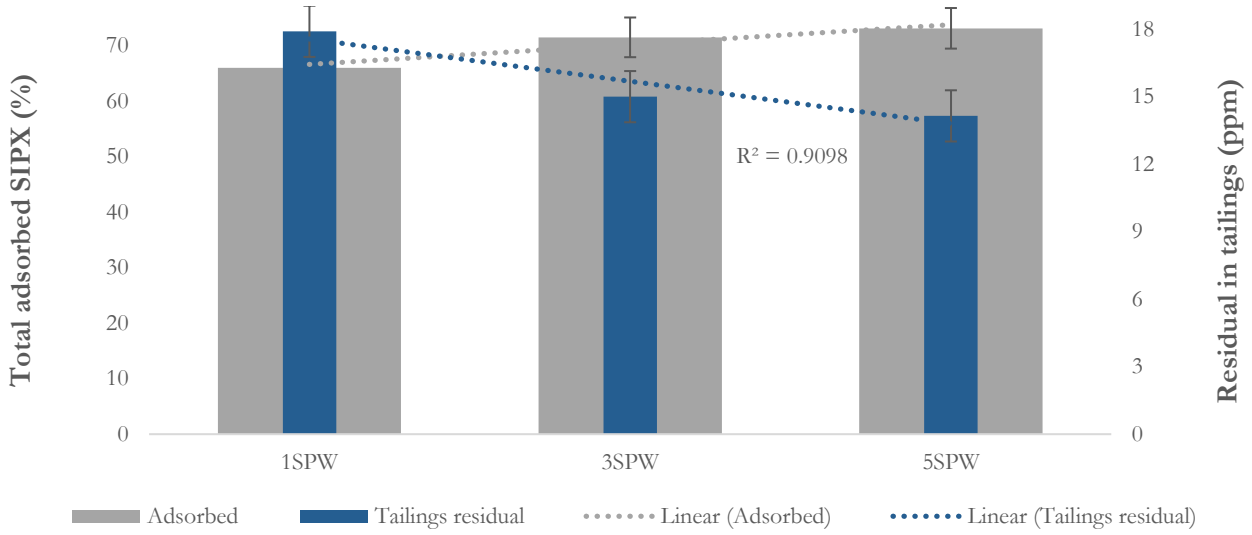

Figure 5.92: A summary of the amount of adsorbed SIPX in mg/litres (=ppm), plotted against the fraction of the initial dosage that was adsorbed in the synthetic plant waters in Run 2

Figure 5.93 plots the amount of SIPX that was adsorbed against the final water recovery in all tested waters, and Figure 5.94 plots the total adsorbed xanthate against the total amount of recovered solids and gangue minerals. Of the actual plant waters, the most amount of collector was adsorbed in raw water, process water and the nickel thickener overflow, but far more water was recovered in raw water when compared to that recovered in process water and the nickel thickener overflow. In fact, though the least amount of xanthate was adsorbed in the copper thickener overflow, it was process water that yielded the lowest water recovery.

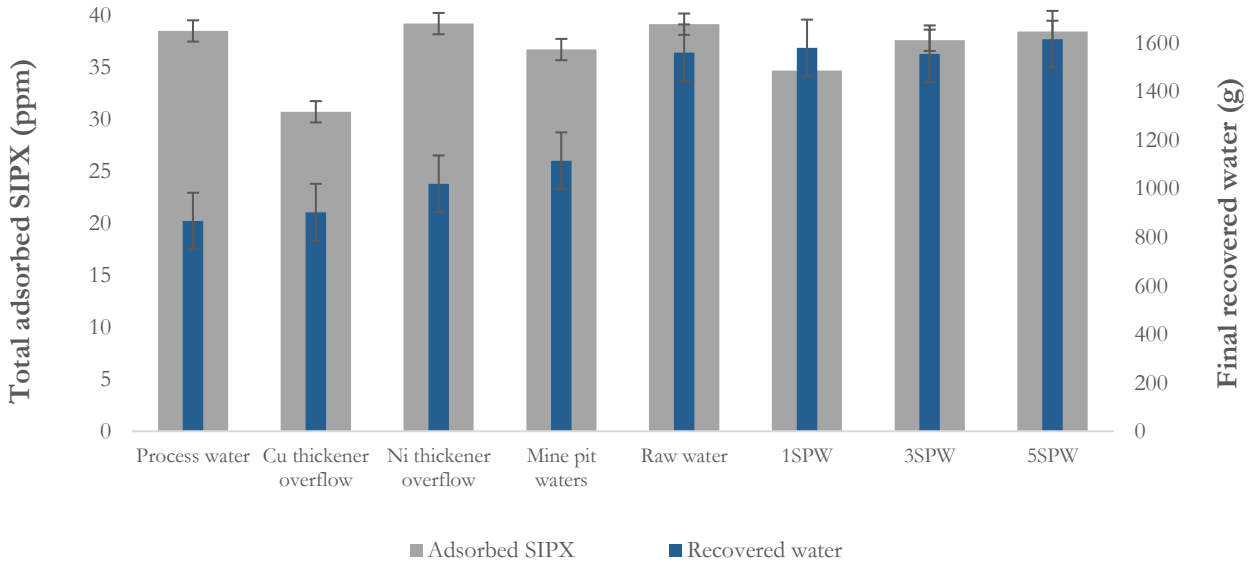

Figure 5.93: Total adsorbed SIPX plotted against final water recovery for Ore B in all tested waters in Run 2

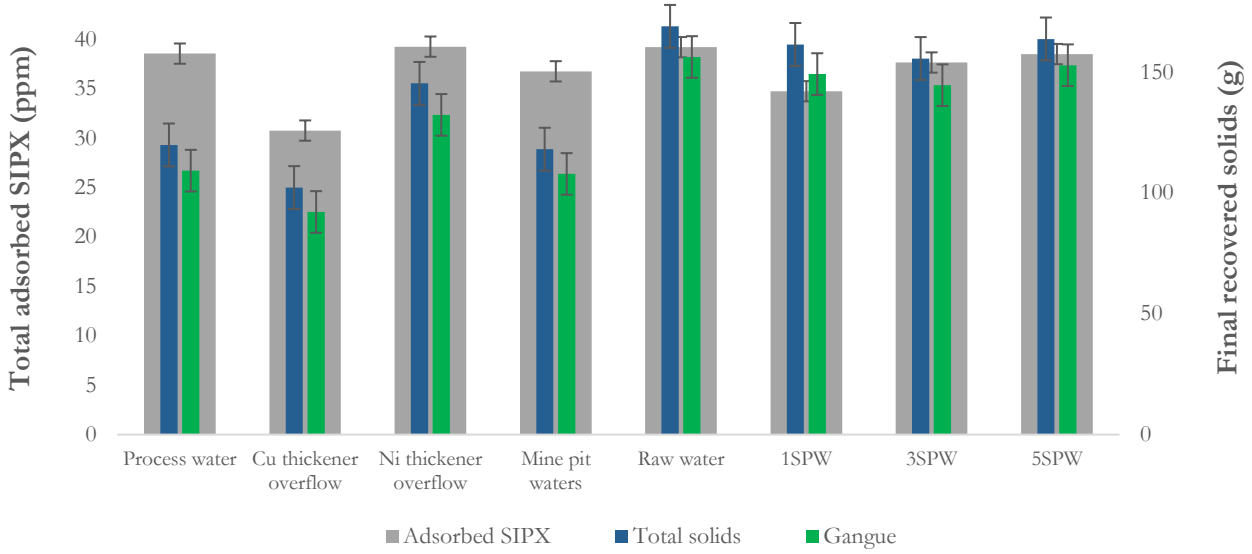

Figure 5.94: Total adsorbed SIPX plotted against final solids recovery for Ore B in all tested waters in Run 2

Of the synthetic plant waters, as the ionic strength of the water increased, so did the amount of total adsorbed SIPX in that water. However, between 1SPW and 3SPW, there isn't much difference in the final water recovery; 5SPW, on the other hand, yielded the highest water recovery. In terms of solids recovery, the synthetic plant waters did not show much difference from each other. Of the actual plant waters, the nickel thickener overflow yielded the lowest solids recovery while the copper thickener overflow yielded the highest.

Figure 5.95 shows the final copper and nickel recoveries against the total amount of adsorbed SIPX., and Figure 5.96 shows the final copper and nickel grades against total amount of adsorbed SIPX.

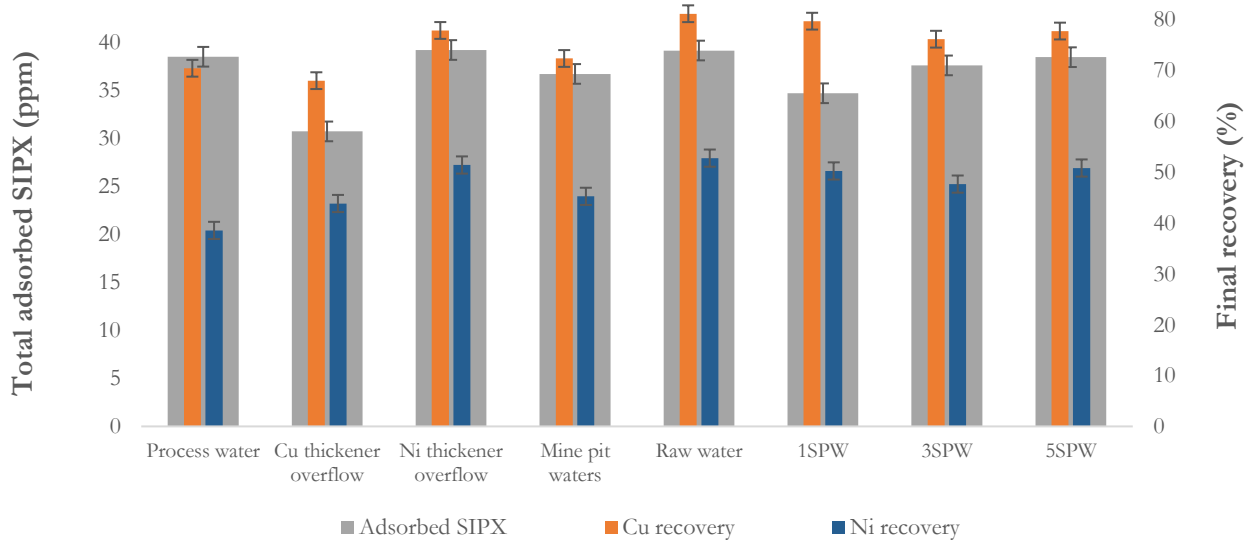

Figure 5.95: Total adsorbed SIPX plotted against final Cu and Ni recovery for all tested waters in Run 2

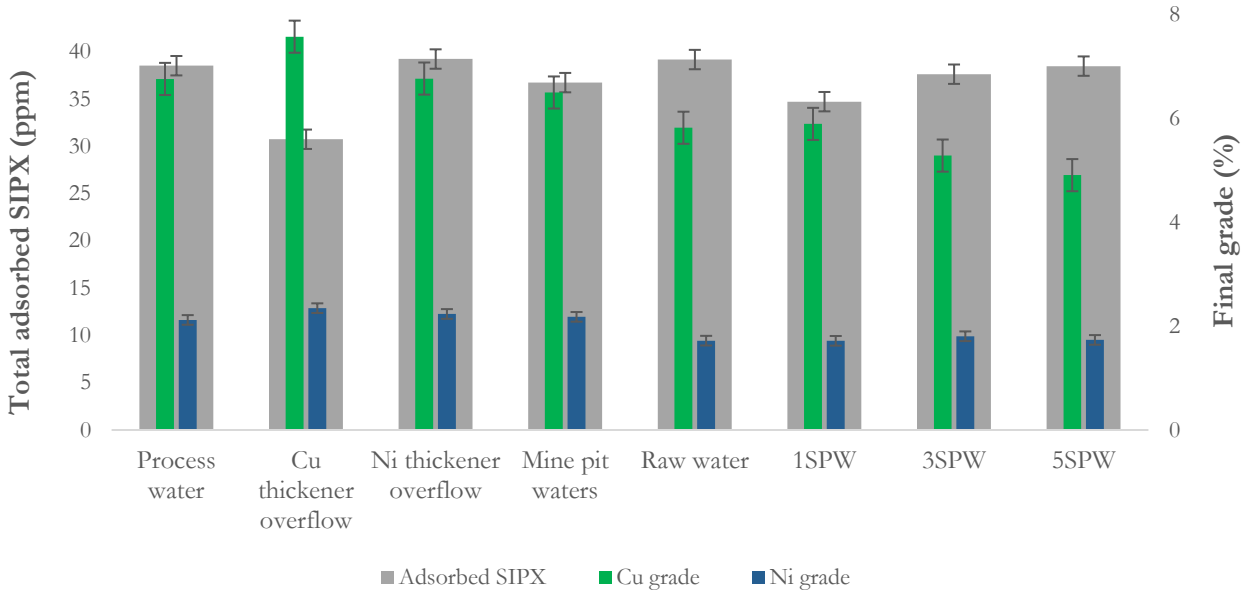

Figure 5.96: Total adsorbed SIPX plotted against final Cu and Ni grade for all tested waters in Run 2

For the synthetic plant waters, although the least amount of xanthate was adsorbed in 1SPW, the highest copper recovery was also observed for 1SPW. But, the highest recovery of nickel was observed for 5SPW, where the highest amount of adsorbed SIPX was also observed. For the actual plant waters, higher adsorption coincided with higher copper recoveries.

In terms of grade, for the synthetic plant waters, as the ionic strength of the water increased, the amount of adsorbed SIPX decreased, but the grade of copper decreased while that of nickel first increased and then decreased. Among actual plant waters, the highest copper and nickel grades were observed when the least amount of SIPX was adsorbed (in the copper thickener overflow).

6. DISCUSSION

The objective of this study was to investigate the effect of water quality on xanthate adsorption in the flotation of two sulphide ores. The flotation performance was evaluated with the following as the key indicators: the recovery of valuable and gangue minerals, and the resultant concentrate grades.

The water quality for this study was defined as the ionic strength in synthetic and actual plant waters (Muzinda and Schreithofer, 2017b; Wiese et al., 2005); in which case, Ore A was floated in synthetic plant waters, and Ore B was floated in both synthetic and actual plant waters. Ore A was floated with SIBX as the sole collector, and Ore B was floated with aerophine and SIPX added sequentially. Of all tested waters, 5SPW had the highest ionic strength at 0.0977 M, and the raw river water had the lowest at 0.0004 M. The amount of collector adsorbed by the mineral particles was measured as that abstracted from solution at given time intervals, and the total adsorbed amount was measured as that abstracted at the end of the flotation period (Hadler et al., 2005; Finkelstein and Lovell, 1972).

In analysing the results, the following is worth noting: after the respective xanthate-conditioning times were concluded, the flotation periods for the ores were identical, at 20 minutes each. However, owing to first being floated in aerophine, Ore B had a total flotation duration of 29 minutes i.e. 9 minutes of aerophine flotation followed by 20 minutes of SIPX flotation. Other primary differences in the respective experimental schemes were as follows: the SIBX was allowed a conditioning time of 2 minutes and the SIPX 1.5 minutes; the SIBX was added to an ore in which none of the extant chalcopyrite had yet been floated, while the SIPX was added to an ore in which most of the chalcopyrite had already been floated, resulting in the xanthates having different target minerals. And of course, the final differences are the respective hydrocarbon chain lengths of the collectors.

Further, as specified in Section 2.5 and Section 4.5.3, owing to variation in the mineralogy of the ores chalcopyrite was used as the key valuable mineral in indicating good flotation performance for Ore A. For Ore B, which is polymetallic, the key valuable minerals were chalcopyrite and pentlandite.

6.1. Ionic Strength Effects on Collector Distribution

The adsorption tests show that overall, regardless of water quality, more collector was abstracted from solution when Ore A was floated. Figure 6.1 is a comparison of the concentration of SIBX and SIPX remaining in 1SPW-Run 1 flotation, expressed as a function of time. Due to the difference in the

volumes of the flotation cells, the 100 g/t collector dosage is equivalent to an initial SIBX and SIPX concentration of 34.5 and 52.6 ppm, respectively. As much as 90% (31.1 ppm) of the initial SIBX was adsorbed 3 minutes after being dosed, while for the same duration, 50% (26.3 ppm) of the SIPX was adsorbed. In each case the rate of adsorption was faster in the first three minutes, after which it slowed and steadied. The differing hydrocarbon chain lengths of the xanthates indicated these results to be aligned with those of Plaksin et al. (1954). That is to say, as far as interacting with the target mineral and hence being abstracted from solution, the collector with the shorter chain was shown to be weaker.

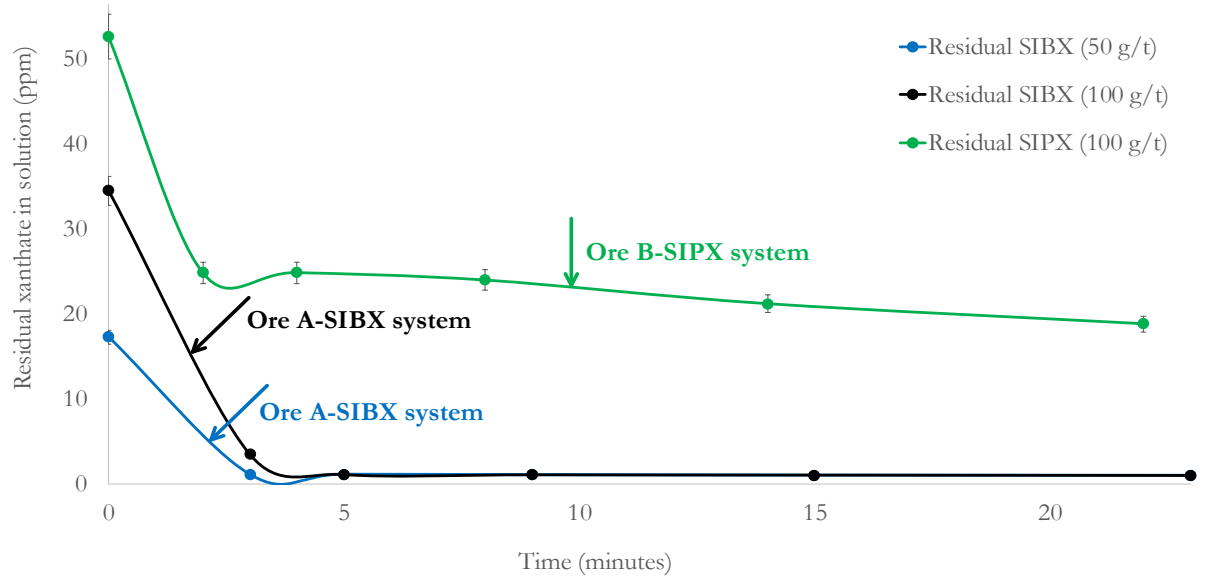

Figure 6.1: Residual concentrations of SIBX and SIPX in 1SPW flotation, and adsorbed fractions of each collector, both expressed as functions of time

The abstraction kinetics agree with those observed by Finkelstein and Lovell (1972) and Hadler et al. (2005). The former demonstrated that in the flotation of phoscorite, as much as 92% of the added fatty acid collector—Unitol DSR, was removed from solution two minutes after being added; afterwards, the adsorption rate was reduced by the accumulation of adsorption products, and the continued adsorption occurred at a constant rate. Hadler et al. found that SIBX adsorption was very rapid in the first thirty seconds of contact with a PGM ore, and that after 2 minutes, 63% of the initial dosage had been adsorbed, following which the adsorption rate steadily decreased.

Moreover, based on the steepness of the slopes in Figure 6.1 (in the first 2 minutes), as the initial concentration of xanthate in solution increased, so did the rapidity of the adsorption kinetics, revealing that the kinetics are a function of initial collector concentration. In fact, doubling the SIBX concentration resulted in nearly double the collector consumption rate in the initial 2-minute duration.

This is best illustrated by Figure 6.2, which simply magnifies Figure 6.1 to show the residual SIBX and SIPX in solution in the first 2 minutes after the addition of each xanthate.

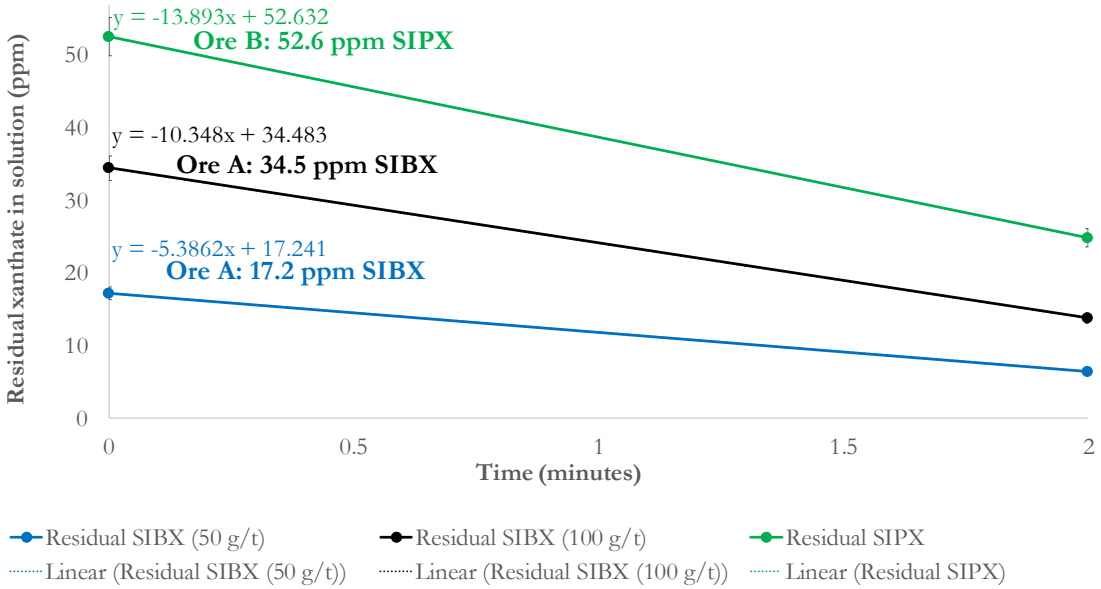

Figure 6.2: Residual SIPX and SIBX in 1SPW flotation as a function of time, as magnified from Figure 6.1

As illustrated in Chapter 5 (Figure 5.16 and Figure 5.17), increasing the synthetic plant water ionic strength in the flotation of Ore A resulted in reduced collector adsorption, while for Ore B, increasing the ionic strength, more notably among the synthetic plant waters, resulted in increased adsorption (Figure 5.75). It is apparent that across the synthetic plant waters, when floating Ore A, the adsorbed amount and adsorbed fraction can each be approximated by a linear relationship. The implications of these results are consolidated with other observed trends in the subsequent sections.

6.2. Ionic Strength Effects on Collector Adsorption and Flotation Response

This section will analyse the results regarding the collector-dosed, non-recycled runs for both ores.

6.2.1. Effects on Flotation Response of a High Grade Copper Ore (Ore A)

For Ore A, the recovery of copper was independent of the amount of adsorbed collector as, for all freshly dosed tests, regardless of the ionic strength and initial SIBX concentration, the final copper recovery was above 94%. Given the study by Kalichini (2015), which showed that treatment of Ore A with 30 g/t SIBX resulted in up to 90% copper recovery, this was an expected result. This is further supported by Hadler et al. (2005), as the authors showed that increasing the collector concentration in the flotation of a PGM ore did not affect the recoveries of the valuable minerals.

As illustrated by Figure 5.23, in each water type, as more SIBX was dosed, and as more SIBX was adsorbed, the grades improved; overall, more SIBX was adsorbed in waters with lower ionic strengths. It is apparent that as more species were dissolved in water, competition between the collector and the dissolved species resulted in less collector being adsorbed (Levay et al., 2001; Shengo et al., 2016), such that flotation in 1SPW resulted in the most collector being abstracted from solution and flotation in 5SPW resulted in the least abstraction.

The final grades were further affected by the total recovery of solids to the concentrate. It was observed that for each tested water, the higher collector dosage and higher adsorption resulted in the reduced recovery of water and solids. This was complemented by increased grades. Similar findings were made by Hadler et al. (2005) and Wang (2016); the authors attributed the improved grades to changes in froth stability. They theorised the following: at lower adsorption densities and particle hydrophobicities, the particles recovered to the froth phase do not affect the froth stability. But as their hydrophobicity increases, froth stability reaches a maximum, after which further hydrophobicity leads to films being ruptured and the increased drainage of water and gangue minerals from the froth.

6.2.2. Effects on Flotation Response of a Nickel-Copper Ore (Ore B)

For the flotation of Ore B in the synthetic plant waters, the degree of collector adsorption did not significantly affect the final recovery of the target pentlandite, and thus did not affect the final nickel recoveries. But for the actual plant waters, higher adsorption coincided with higher nickel recoveries.

This contrast to the results observed for Ore A suggests that collector adsorption is not only dependent on water quality, but also on the mineralogy of each ore, and how the various minerals interact with each other and therefore influence each other's floatability. As outlined in Chapter 2, the mineralogies of these two ores are vastly different from each other, with the valuable chalcopyrite comprising 3.9% of Ore A, and the combined chalcopyrite and pentlandite comprising only nearly 2% of Ore B. Additionally, pentlandite naturally floats slower than chalcopyrite (Corin et al., 2011; Gray et al., 2016; Rao, 2000). This is demonstrated by the collectorless runs, where, without enhanced hydrophobicity, the final copper recoveries were consistently higher than the final nickel recoveries.

Among the synthetic plant waters, the highest nickel grades were achieved for the most adsorbed collector i.e. in 3SPW, followed by 5SPW. For the actual plant waters, the highest adsorption (in raw water and the nickel thickener overflow) did not coincide with the highest grade (in process water). Process water has a higher ionic strength than both the nickel thickener overflow and raw water. But

overall, no apparent relationship emerged between the ionic strength of the actual plant waters and collector adsorption. This suggests that for some waters, factors such as the respective natures and concentrations of the dissolved ions, or combinations of ions, are possibly more effective than the overall conductivity of the water. This can be gleaned from Ikumapayi et al. (2012), Shengo et al. (2016) and Kirjavainen et al. (2002), who showed that different ions and different concentrations of these ions have varying effects on collector adsorption and overall flotation.

In contrast, the synthetic plant waters are easier to control since their nature allows for the manipulation of the overall ionic strength by multiplying, in known quantities, the presence of known species. And as such, the overall linearity of the relationship between synthetic plant water ionic strength and collector adsorption is, perhaps, an easier one to assert.

Though the relationship between the ionic strength of the actual plant waters and collector adsorption is tenuous, it was observed that as water recovery increased, so did the recoveries of the total solids and the valuable minerals. Kawatra (2009) and Senior et al. (1995) postulate that the increased recovery of water into the froth results in increased particle entrainment. By this logic, it stands to reason that in addition to xanthate-induced flotation, entrainment contributed to the total solids as well as nickel recoveries. Raw water flotation yielded the highest water, solids and valuable mineral recoveries.

Figure 6.3 compares the total water and solids recovered in the synthetic plant water flotation of the two ores. The figure shows that overall, more water was recovered when floating Ore B, while more solids were recovered when floating Ore A.

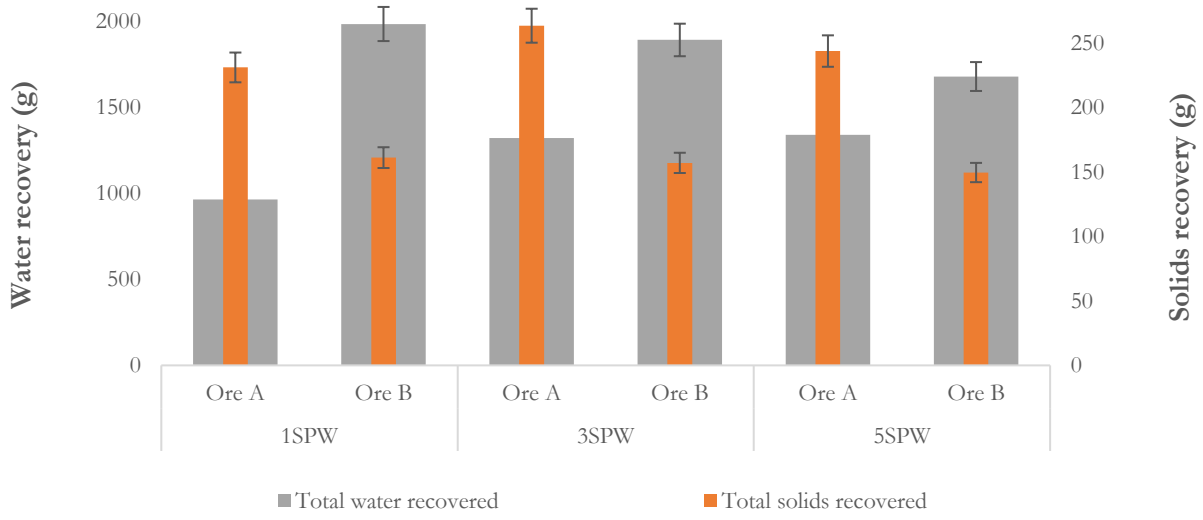

Figure 6.3: A comparison of the total water and solids recovered to the concentrate for the two ores

This, coupled with the overall higher adsorption in Ore A flotation, suggests that the respective gangue compositions influenced entrainment, and that entrainment was likely a bigger factor in Ore B flotation. This is supported by the solids per unit water recoveries, exemplified by Figure 6.4, which shows the non-cumulative solids per unit mass recovered in the first and in the fourth concentrates i.e. C1 and C4. By increasing the depressant (guar) dosage to suppress the recovery of naturally floatable gangue in a Merensky ore, and by observing that at 500 g/t, no naturally floatable gangue was recovered, Wiese (2009) has determined an entrainment factor from a linear relationship between the solids recovered per unit water. Figure 6.4 shows an increase in the recovered solids per unit water mass, with Ore A recoveries being consistently higher than those observed for Ore B.

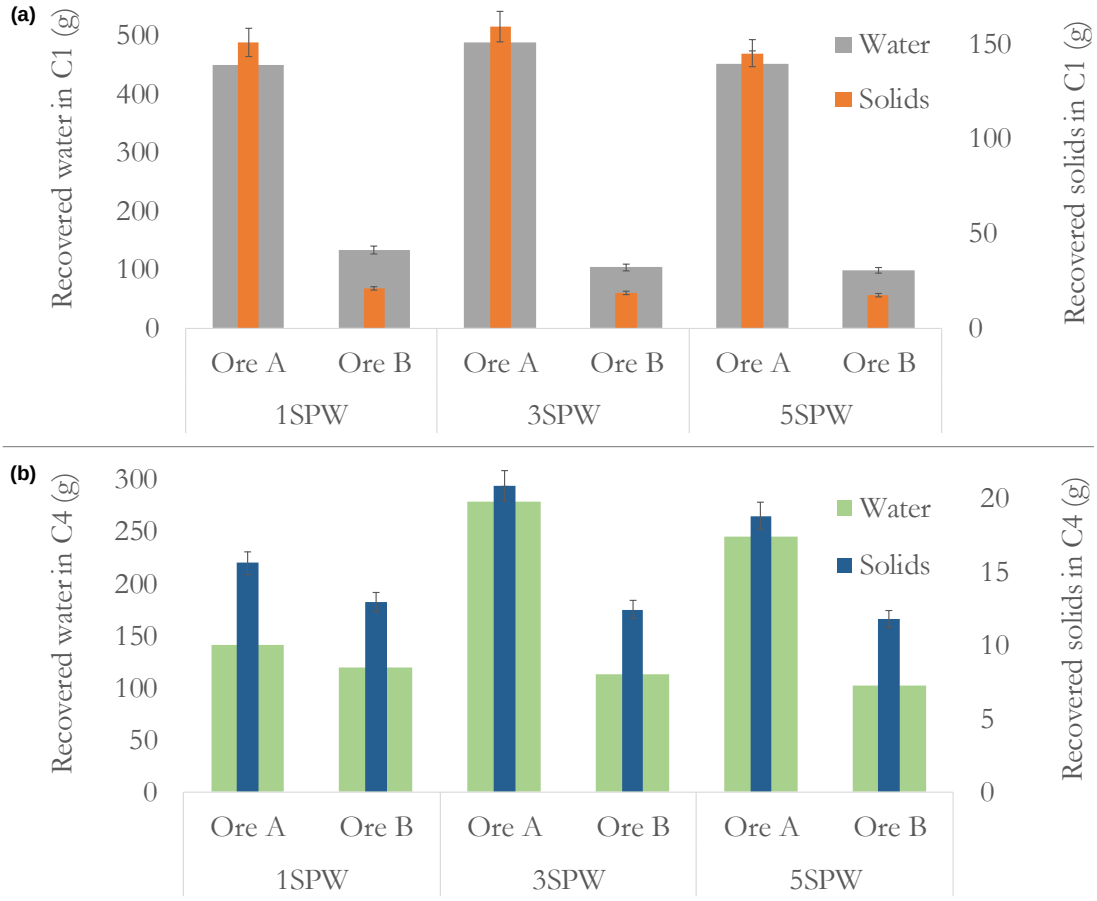

Figure 6.4: A comparison of the recovered solids per unit water; (a) non-cumulative masses recovered in C1, (b) non-cumulative masses recovered in C4

The large differences in the final recovered solids further suggest that though more SIBX than SIPX was adsorbed, the SIBX was less selective, or that gangue materials associated with Ore A are more floatable, in a manner similar to findings made by Wiese et al. (2006). This is considering that the total

flotation period for Ore B was longer by 9 minutes, thus providing more time for solids to report to the concentrate by either true flotation or entrainment (Senior et al., 1995). The most ubiquitous gangue minerals in Ore A are plagioclase-feldspar (32.7%), mica (23.3%) and quartz (25.4%). The most ubiquitous gangue minerals in ore B are amphibole-hornblende (33.0%) and diopside (38.6%). In a study investigating the effect of copper activation and collector adsorption on gangue minerals, Martinovic et al. (2004) showed that the floatability of minerals such as quartz and feldspar could be significantly improved by inadvertent adsorption.

Moreover, among others, studies by Lotter et al. (2003) and Pietrobon et al. (1997) have shown that there exists a correlation between the presence of magnesium-bearing silicates and poor pentlandite flotation. They suggest that the magnesium silicates form slime coatings on pentlandite surfaces, resulting in reduced collector adsorption and hence decreased pentlandite flotation. It is worth noting that the dominant gangue minerals in Ore B i.e. diopside and amphibole-hornblende, are both magnesium-bearing silicates. In addition to that, the sulphide content of Ore A (~8.1%) is higher than that of Ore B (~5%); sulphides are weakly polar and tend to interact poorly with water (Wills and Napier-Munn, 2006; Wrobel, 1970), hence, the naturally hydrophobic fraction of Ore A was higher than that of ore B. And further, the sulphides in Ore B are highly disseminated; in the case of partial liberation, or no liberation at all, the naturally hydrophobic fraction of Ore B was likely further reduced.

6.3. The Effect of Initial Collector Dosage

Increasing the SIBX dosage from 50 to 100 g/t in the flotation of Ore A resulted in decreased water and solids recoveries, and increased concentrate grades. In addition to the factors already discussed, as suggested by Kirjavainen et al. (2002), Shengo et al. (2016) and Hadler et al. (2005), the improvement in grades is either a result of increased valuable mineral recovery or reduced gangue mineral recovery. But the recovery of copper was unaffected by initial collector dosage, implying that the improved grades were a result of reduced gangue loading in the froth phase.

Considering the total water and solids recovery, it is apparent that froth stability was affected by collector dosage, or more specifically, froth stability was affected by the hydrophobicity of the particles present in the froth. Hydrophobic particles accelerate the rate of liquid drainage, thus increased concentrations of hydrophobic particles (as a result of increased collector dosage) in the froth reduce the recovery of hydrophilic particles and water in the concentrate. Water recovery is an indicator of froth stability (Manono et al., 2013; Tao et al., 2000), and entrainment is a function of water recovery

(Kawatra, 2009). By these definitions, it stands to reason that at 100 g/t SIBX, when floated particles were highly hydrophobic, the instability of the froth (as compared to that achieved at 50 g/t) resulted in increased drainage of entrained materials. And given the improved grades, it stands to reason that a substantial fraction of the entrained materials were gangue minerals.

6.4. The Effect of Recycled Xanthate

This section is divided into two subsections. The first, 6.4.1, will analyse the results pertaining to the recycle runs in which, in addition to the residual collector recycled from Run 1, fresh collector was dosed. Subsection 6.4.2 will analyse the results pertaining to the recycle runs in which collector was not re-dosed, and only the residual collector recycled from Run 1 was relied on for enhanced mineral hydrophobicity.

6.4.1. The Effect of Recycled Xanthate in Conjunction with a Fresh Dosage

Given that more collector was adsorbed in Ore A flotation, it was the case that overall, higher xanthate concentrations were recycled in Ore B flotation.

6.4.1.1. Effects on the Flotation Response of a High Grade Copper Ore (Ore A)

Two trends were observed when Ore A was floated in recycled waters (Figure 5.11, Figure 5.26 and Figure 5.41 in Chapter 5). For the 50 g/t SIBX batch, when compared to their non-recycled counterparts, the recycled runs yielded lower solids and water recoveries. While for the 100 g/t SIBX batch, when compared to their non-recycled counterparts, the recycled runs yielded higher solids and water recoveries. Additionally, for the 50 g/t SIBX batch, higher grades were observed in the recycled runs; while for the 100 g/t SIBX batch, lower grades were observed in the recycled runs (Figure 5.15, Figure 5.30 and Figure 5.45).

Since neither the SIBX dosage nor the recycle simulations affected copper recovery, the findings suggest that, similarly to that discussed in Section 6.3, the concentrate grades were controlled by the recovery of gangue minerals. That is to say, in the case of the 50 g/t SIBX batch, the improved grades in the recycled runs were a result of reduced gangue loading in the froth phase; and in the case of the 100 g/t SIBX batch, the reduced grades in the recycled runs were a result of increased gangue loading in the froth phase (Hadler et al., 2005; Kirjavainen et al., 2002; Shengo et al., 2016).

All recycled tests showed higher adsorption values than their non-recycled counterparts; this, as a result of recycled residual xanthate, and hence higher SIBX concentrations being present in solution

and available for abstraction. In addition, increasing the SIBX dosage from 50 to 100 g/t resulted in decreased water and solids recoveries, and increased concentrate grades. Thus, the observed trends were typical in that, increased collector adsorption and mineral hydrophobicity reduced the froth stability and yielded lower water and solids recoveries, and since the copper recoveries were unaffected by these factors, higher concentrate grades were obtained across the two SIBX dosages.

6.4.1.2. Effects on the Flotation Response of a Nickel-Copper Ore (Ore B)

In the case of the polymetallic Ore B, the recycled SIPX did not only affect the flotation response of the target mineral—pentlandite, but also the response of chalcopyrite. Figure 6.5 illustrates the comparison of the copper recoveries and grades in the first 9 minutes of flotation, in Run 1 (with aerophine present and no SIPX) and Run 2 (with aerophine and residual SIPX). The figure shows that for all waters, the flotation response of chalcopyrite was hindered in the presence of residual xanthate since, in the recycle runs (Run 2), reduced copper recoveries and grades were observed for all water types, implying that in this case the effect of residual xanthate is stronger than that of water quality.

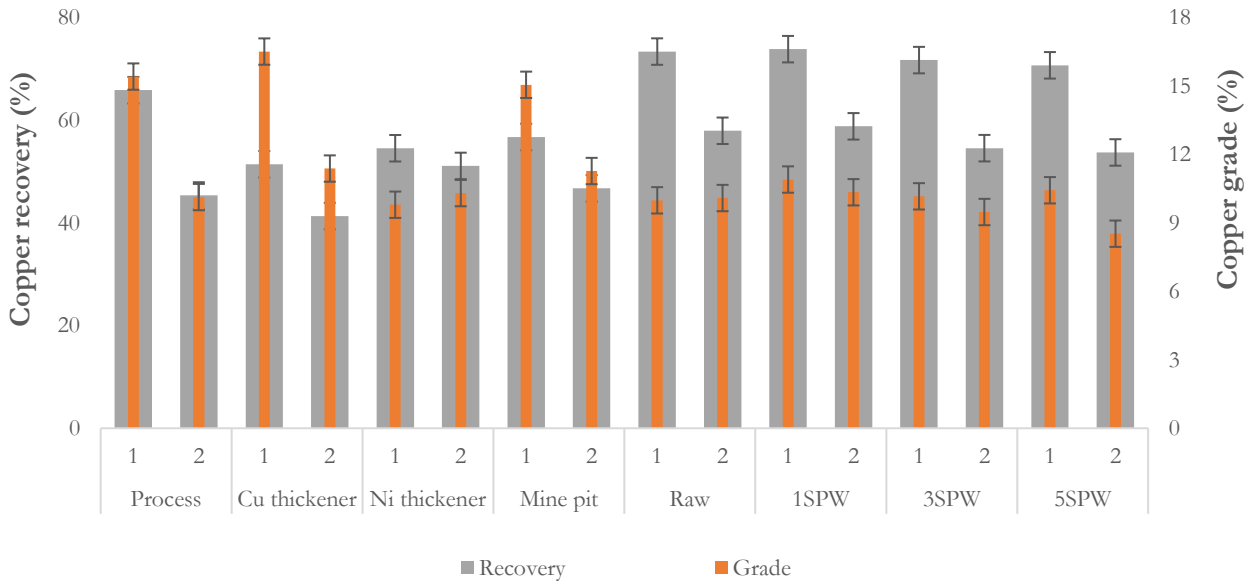

Figure 6.5: A comparison of the copper recovery and grade in Run 1 (no added SIPX) and Run 2 (residual SIPX present), in the first nine minutes of Ore B flotation

The loss in copper recovery was likely a result of the following: aerophine, though not as strong a collector as SIPX, is preferred for its selectivity, and thus preferred for the flotation of such fast-floating minerals as chalcopyrite (Gray et al., 2016; Cytec Solvay Group, 2018). As a result, the non-recycled runs yielded higher copper recoveries because the dosed collector was selective. SIPX, being stronger and less selective (Bulatovic, 2007; Muzenda, 2010), in the recycle runs, likely adsorbed on

the minerals indiscriminately. As an alternative, or perhaps compounding this, is the likelihood that in the presence of two collectors (one recycled and another freshly dosed), competition arose between the collectors, resulting in reduced efficiency of the weaker aerophine (Levay et al., 2001)

Figure 6.6 illustrates the comparison of the nickel recoveries and grades in the first 9 minutes of flotation, in Run 1 i.e. the non-recycled run (with aerophine present and no added SIPX) and Run 2 i.e. the recycle run (with aerophine and residual SIPX present). The 9-minute mark is where fresh SIPX was added to the cell.

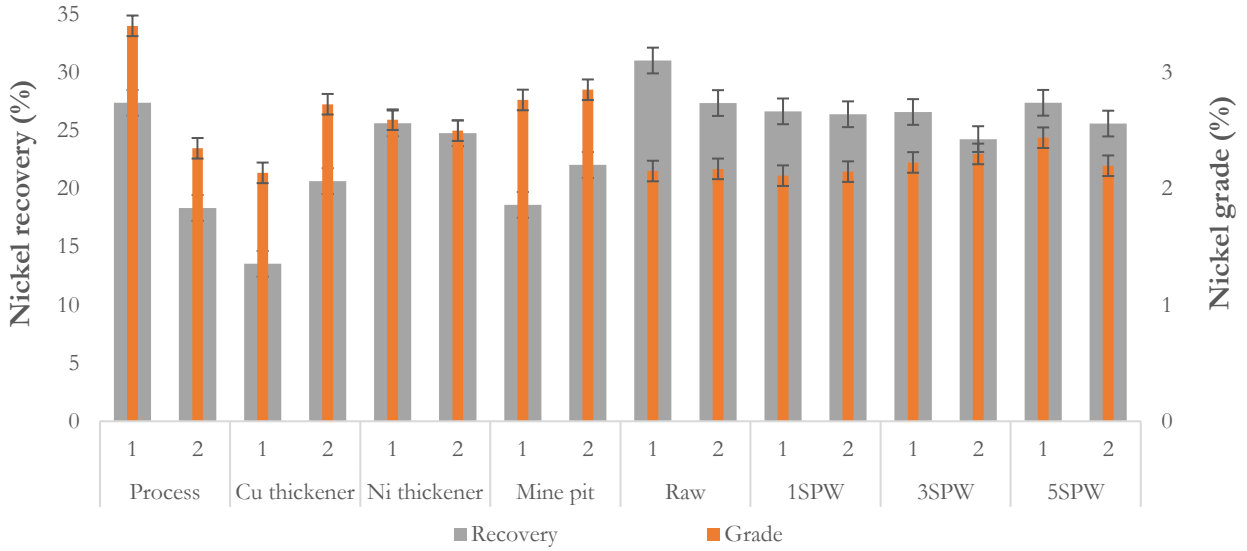

Figure 6.6: A comparison of the nickel recovery and grade in Run 1 (no added SIPX) and Run 2 (residual SIPX present), in the first nine minutes of Ore B flotation

Process water, raw water and the nickel thickener overflow commenced the recycle runs with the least amount of collector when compared to the other waters. In the recycle scenarios, despite more SIPX being adsorbed in these three waters, the nickel recoveries were reduced when compared to the non-recycled runs. On the other hand, waters like 1SPW, the mine pit waters, and the copper thickener overflow, which commenced the recycle runs with the three highest SIPX concentrations, all yielded higher nickel recoveries and grades when compared to their non-recycled counterparts. And as with the non-recycled tests, the least SIPX was adsorbed in the copper thickener overflow.

For these scenarios, the recoveries of water and solids as functions of time were as follows: in the first 9 minutes, both parameters were less than those recovered in the recycled runs, but in the last 20 minutes (after more SIPX was added), both the water and solids recoveries in the recycled runs were more than those obtained in the non-recycled runs. For the full 29 minutes, more water was recovered

in non-recycled tests while more solids were recovered in the recycled runs. The gangue mineral recoveries were increased, suggesting reduced collector selectivity in all waters.

The recovery profile can thus be summarised as follows: the presence of recycled SIPX in the different waters resulted in non-selective hydrophobic coating (Shengo et al., 2016), further resulting in the increased loading of hydrophobic particles to the froth, where ruptured films reduced the recovery of water and entrained materials (Kawatra, 2009). Additionally, as a possible result of the presence of recycled SIPX in the mill, freshly liberated pentlandite surfaces contacted the collector before oxidising, and the metallurgical performance was improved, similar to findings made by Wiese et al. (2006). Figure 6.7 summarises the metallurgical performance of Ore B by comparing the final copper recoveries and grades obtained between the recycled (Run 1) and non-recycled (Run 2) tests.

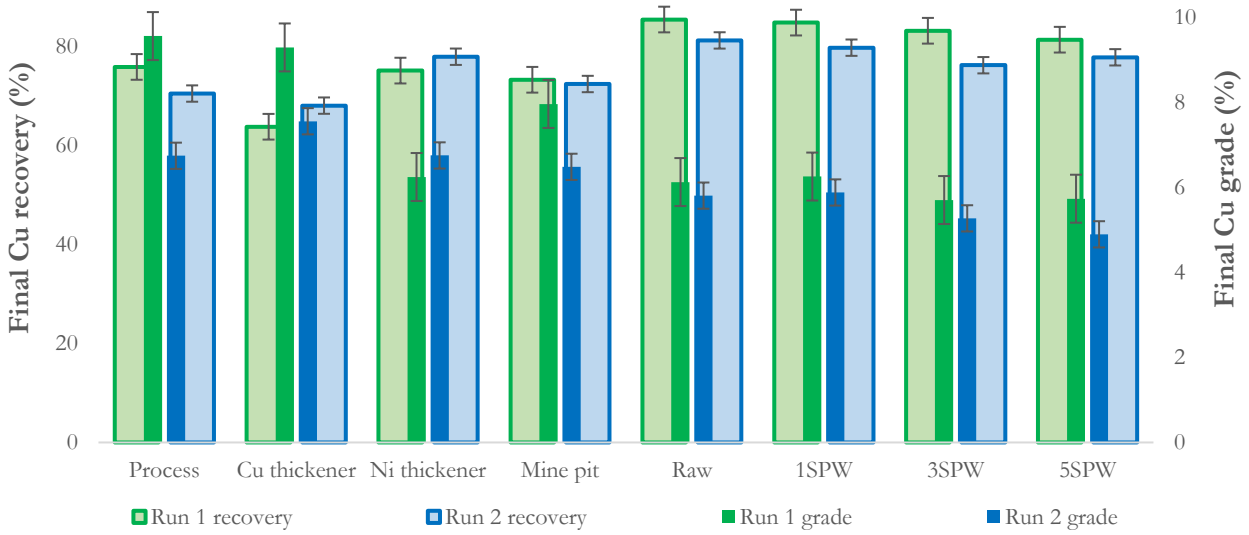

Figure 6.7: A comparison of the final copper recoveries and grades obtained between Run 1 (flotation in non-recycled water) and Run 2 (flotation in recycled water) for the full flotation period

At the end of the full flotation period, the recycle runs yielded the lowest overall copper recoveries, indicating that Ore B was more sensitive to changes in the initial collector concentration; this, likely due to Ore B's polymetallic nature. When floating a polymetallic ore in a locked-cycle simulation similar to that employed in this study, Shengo et al. (2016) found that the more naturally floatable mineral (malachite) was more negatively affected than the less naturally floatable mineral (heterogenite). By floating the ore in deionised water contaminated with known concentrations of bicarbonate, magnesium, sulphate and thiosulphate ions, the authors showed that the reduced flotation efficiency was due to the accumulation of chemical species in the feed water. It was theorised that these species enhanced the hydrophilicity of malachite but at the same time, enhanced the

hydrophobicity of heterogenite. Thus, in a manner similar to that postulated by Senior et al. (1995), the floatability of one mineral resulted in the reduced floatability of another.

Moreover, certain ions, and certain concentrations of ions have been shown to favour the floatability of some minerals while depressing the flotation of others. For example, Ruan et al. (2018) have shown that moderate concentrations of Ca^{2+} and Mg^{2+} improve the flotation of apatite, but have an insignificant effect on the flotation of dolomite. The authors also showed Al^{3+}, Fe^{3+} and excessive concentrations of Ca^{2+} to decrease apatite and dolomite recovery. This was attributed to the adsorption of the metal ions on the mineral surfaces. Calcium ions were shown to enhance the adsorption of anionic collectors on the apatite surfaces, and to leave the dolomite surfaces unchanged.

In another study, Bulut and Yenial (2016) showed the floatability of galena and sphalerite to be favoured by recycled water and increased ion concentrations, while pyrite was depressed in these conditions. However, beyond a certain ion concentration, the galena was also depressed. And again, the presence and concentrations of metal ions (Ca^{2+}, Mg^{2+} and SO_4^{2-}), and the adsorption of these ions onto the mineral surfaces was concluded as the cause for such differing responses.

Of the three investigated systems (collectorless, non-recycled and recycled), the recycle simulations yielded the highest overall nickel recoveries, indicating the propensity of pentlandite to float better in the presence of a stronger collector (SIPX), and at higher concentrations of the collector. This is summarised by Figure 6.8, which shows a comparison of the final nickel recoveries and grades obtained between the non-recycled (Run 1) and recycled (Run 2) tests.

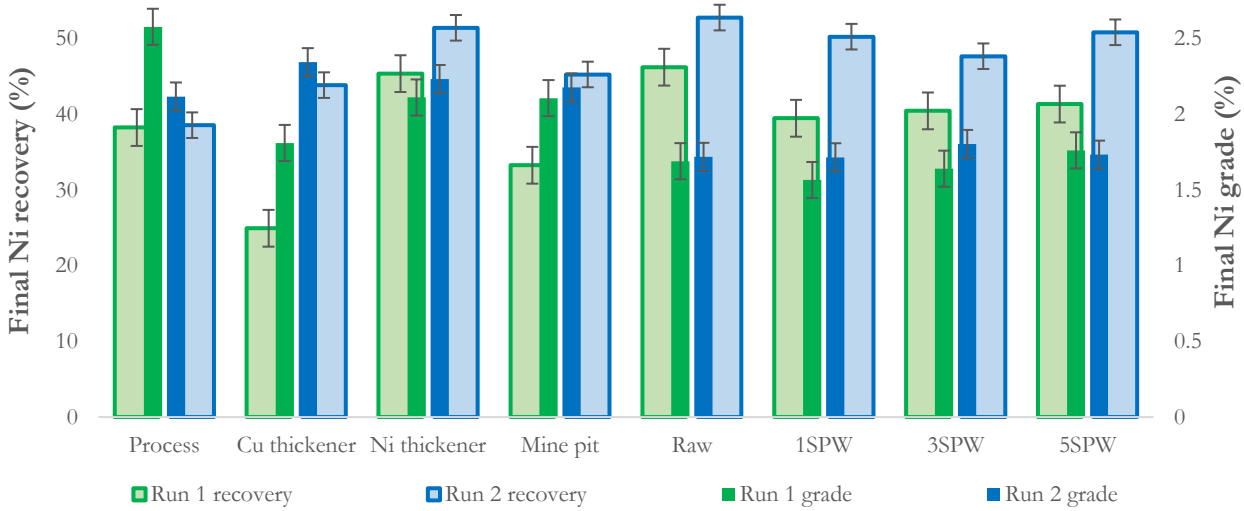

Figure 6.8: A comparison of the final nickel recoveries and grades obtained between Run 1 (flotation in non-recycled water) and Run 2 (flotation in recycled water) for the full flotation period

6.4.2. The Effect of Recycled Xanthate without Fresh Dosage

The results discussed in this section pertain to the flotation of Ore A in recycled water and no additional SIBX. In the first recycle (i.e. Run 2) the water recoveries in all cases were higher than those yielded in the collectorless and the non-recycled runs, and higher than those yielded in collector-reloaded recycle runs. This was true for each water type. And as with the previous runs, increasing the ionic strength resulted in increased water recovery. However, the mass of recovered solids was less in these recycle runs than the non-recycle, and each water type recovered less solids than in its re-dosed counterpart. For example, though 1SPW-*50,0* flotation yielded more water that 1SPW-*50,50*, 1SPW-*50,0* also yielded less solids than 1SPW-*50,50*. At the same time, 1SPW-*50,50* yielded a higher copper recovery than 1SPW-*50,0*, as was the case for all waters and dosages, as summarised in Figure 6.9.

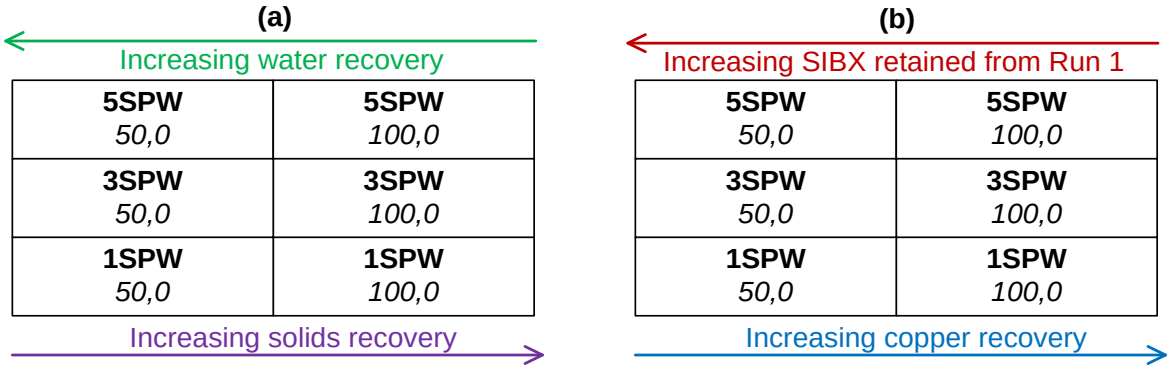

Figure 6.9: The trend of copper recovery for Ore A floated solely in recycled SIBX; (a) the summary of water solids and water recoveries, (b) the summary of recycled SIBX concentrations and copper recoveries

An increase in the concentration of recycled SIBX was contrasted by a decrease in copper recovery. At the same time, the increase in recycled SIBX was complemented by an increase in the recovery of gangue minerals. Shengo et al. (2016) have shown that the locked-cycle recycling of synthetic process water increases the concentration of dissolved alkaline earth metals in solution, and that this limits the hydration of gangue materials, and therefore increases the recovery of these materials.

The once-recycled experiments (Run 2) yielded higher recoveries and grades than the collectorless runs. For example, collectorless 1SPW yielded a copper recovery of 42.5%, while *50,0* and *100,0* 1SPW yielded respective 62.6 and 82.3% copper recoveries. This demonstrates that even with minute initial quantities of SIBX, chalcopyrite was highly floatable; and in the case of once-recycled 1SPW-*100,0*, requiring as little as 1.68 g/t SIBX to yield 82.3% copper recovery. Overall, this suggests that in the case of Ore A, the metallurgical performance of chalcopyrite is favoured, at the very least, by recycling

synthetic process water. The elemental sulphur formation responsible for the collectorless flotation of chalcopyrite (Bowden and Young, 2016; Kelebek et al., 1996), coupled with the adsorption of extant SIBX, renders the chalcopyrite sufficiently hydrophobic to float.

This is supported by Figure 6.10, which summarises the metallurgical performance of Ore A when floated in 1SPW and 0.5 g/t SIBX. The figure shows that even at this slight dosage, chalcopyrite is rendered sufficiently hydrophobic to float better than is the case in a collectorless system, yielding a final copper recovery of 65.7% and grade of 4.89%.

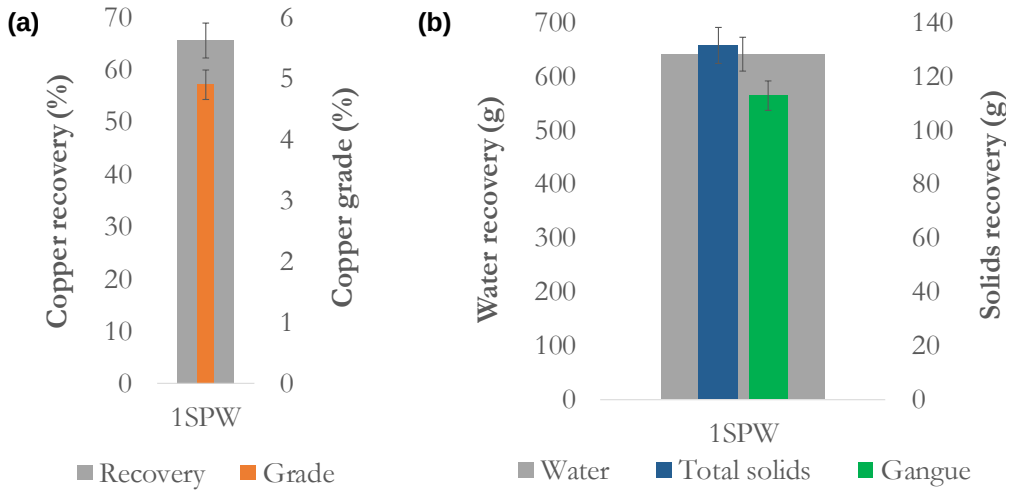

Figure 6.10: A summary of the metallurgical performance of Ore A when floated in 1SPW and 0.5 g/t SIBX; (a) the final achieved copper recovery and grade, (b) the final recovered water, solids and gangue materials

In the case of twice recycled waters (Run 3), it was theorised that given the proven high adsorption of SIBX onto Ore A, and given that very little collector was recycled from Run 1 (non-recycled waters) to Run 2 (once-recycled waters), no collector remained at the end of Run 2 and therefore no collector was recycled to Run 3. The froth phase observed in Run 2 was excessive such that the water recoveries exceeded those yielded in Run 1, and even those yielded in the collectorless runs. Thus, a third run in which neither collector nor frother was dosed, was conducted.

This was done in order to gauge whether other residual species from previous runs, be it the frother or the dissolved ions, could sufficiently influence the metallurgical performance of the fast-floating ore even in the absence of collector. No positive influence was observed as the copper recoveries were low such that they were less than those yielded in the collectorless experiments, being as low as 18% and 19% in the case of 3SPW-*50,0* and 3SPW-*100,0*, respectively.

The combination of factors resulting in the poor recovery can be summarised thusly: the mineral particles were not rendered sufficiently hydrophobic, and though the collectorless runs previously showed copper recoveries as high as 67%, the presence of frother, and hence an actively controlled froth that allowed for the separation of chalcopyrite from the pulp slurry (Finch, 1995), proved essential in the collectorless cases.

7. CONCLUSIONS

The objective of this study was to investigate the effect of water quality on xanthate adsorption in the flotation of two sulphide ores. In order to determine the relationship between key performance indicators and the factors influencing them, five key questions, as outlined in Chapter 3, were formulated. It was in answering these that the validity of the postulated hypothesis could be examined.

1. **How does increasing plant water ionic strength affect the collector adsorption in the flotation of a sulphide ore?**

The study showed that collector adsorption is not only dependent on water quality, but also on the mineralogy of each ore.

a) Effects on a Kansanshi High Grade Copper Ore (Ore A)

The linear relationship between the concentrations of specific, known ions in the different synthetic plant waters allowed for a perceivable trend on the influence of ionic strength on collector adsorption. That is to say, for waters that have identical ionic species, and with these species varied in known concentrations, the relationship between water quality and collector adsorption is easier to predict than in cases where only one overall parameter (ionic strength) is definitively known. And in the case of synthetic plant waters, increasing the ionic strength of the water, i.e. reducing the water purity, resulted in decreased xanthate adsorption. And so, for Ore A, which was floated in waters whose chemical compositions were more fully known, the recovery of copper was unaffected by water quality and adsorption, while the concentrate grades were increased with increasing collector adsorption.

b) Effects on a Lapland Nickel-Copper Ore (Ore B)

For the polymetallic Ore B, for which the properties of the used waters were less known, overall, no apparent relationship emerged between the ionic strength of the actual plant waters and collector adsorption. However, for the synthetic plant waters, though the degree of collector adsorption improved the final nickel grades, it did not significantly affect the final nickel recoveries. But for the actual plant waters, higher adsorption coincided with higher nickel recoveries, but lower grades.

2. **How does increasing plant water ionic strength affect the collector distribution in the flotation of a Kansanshi copper ore and a Lapland nickel-copper ore?**

Overall, regardless of water quality, more SIBX than SIPX was abstracted from solution. The adsorption kinetics were such that for all waters, three minutes after being added to the cell, as much as 85% of the initial SIBX was adsorbed, and for the same duration, 51% of the SIPX was adsorbed; after which, for both systems, the rate of adsorption slowed and steadied. As a result, the bulk of the total adsorbed collector was adsorbed in the first three minutes. For the synthetic plant waters, increasing the ionic strength resulted in decreased adsorption kinetics, while for the actual plant waters, no emerging trend was observed. Moreover, as the initial concentration of xanthate in solution increased, so did the rapidity of the adsorption kinetics.

3. **How does initial collector dosage affect the flotation response of a Kansanshi copper ore? What are the adsorption profiles of varying initial collector dosages?**

Increasing the SIBX dosage in the flotation of Ore A did not significantly affect the copper recovery, but the grades were improved. As already stated, increasing the concentration of SIBX resulted in faster adsorption kinetics. The adsorption profiles were such that increasing the initial concentration of SIBX resulted in more of the collector being adsorbed and thus less of it remaining in solution.

4. **How does recycling residual collector affect the flotation response of a sulphide ore?**

    a) Effects on a Kansanshi High Grade Copper Ore (Ore A)

Floating Ore A in waters containing recycled SIBX did not affect the recovery of copper. However, higher concentrate grades were observed for recycled waters with lower SIBX concentrations, while lower grades were observed for recycled waters with high SIBX concentrations. Additionally, the ore was highly floatable such that even minute dosages of 0.5 g/t SIBX (in 1SPW) render the valuable chalcopyrite sufficiently hydrophobic to yield a copper recovery of 65.7%. The study showed that increasing the concentration of recycled SIBX, and floating the ore without re-dosing additional SIBX, resulted in decreased copper recoveries and grades.

    b) Effects on a Lapland Nickel-Copper Ore (Ore B)

Floating Ore B in waters containing recycled SIPX, regardless of water quality, resulted in reduced copper recoveries and grades. In contrast, depending on the water, the flotation response of pentlandite in the presence of recycled SIPX was improved, as indicated by nickel recoveries and grades obtained for non-recycled and recycled runs. This was such that the highest overall copper

recoveries were yielded in non-recycled raw water, 1SPW and 3SPW; and the highest overall grades were yielded for non-recycled process water and the copper thickener overflow. Overall, higher nickel recoveries and grades were yielded in recycled waters.

5. **Which, between the collector dosage and water quality, has a greater effect on the flotation response of a Kansanshi high grade copper ore and a Lapland nickel-copper ore? Can their respective effects be decoupled?**

The study has shown that water quality influences the extent of xanthate adsorption, such that certain waters, such as the copper thickener overflow, are less conducive to adsorption than waters such as the raw and standard process. The study has also shown that in the case that the dissolved ionic species are identical, increasing the ionic strength of water yields a linear decrease in xanthate adsorption. However, increased adsorption did not necessarily result in improved grades and recoveries; and further, it was shown that other factors such as ore mineralogy and xanthate chain length contribute to overall metallurgical performance. Thus, the respective effects of collector dosage and water quality could not be definitively decoupled.

8. RECOMMENDATIONS

Following the results and conclusions, the following recommendations are made:

- In the case of the actual plant waters, a complete chemical analysis should be conducted, following which, flotation in varying concentrations of single ions can be conducted to quantify the degree of influence of the ions.
- Additional collector dosages should be investigated, especially for Ore B since it proved more sensitive to initial xanthate concentrations. As an addendum, additional recycle runs should be simulated, both with and without a re-dosage of SIPX.
- The model systems should be expanded such that Ore A is floated in SIPX, and Ore B is floated in SIBX, and perhaps other xanthates too in order to more accurately quantify the extent of hydrocarbon chain length influence.
- A method should be developed to monitor the adsorption profile of aerophine similarly to the xanthates. As an addendum, a study in which Ore A is floated in fresh and recycled aerophine should be conducted.
- A study encompassing other reagents should be conducted.

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APPENDICES

Appendix A: Kansanshi Ore Flotation and Adsorption Tests

A.1. Distilled Water at 0, 50 and 100 g/t SIBX

Where: C = concentrate weight (g)

B = wash bottle weight (g)

$B + H_2O$ = full wash bottle weight (g)

D = concentrate dish weight (g)

Weight of parameter (g)	Distilled water at 0 SIBX				Distilled water at 50 SIBX				Distilled water at 100 SIBX											
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	83.38	41.5	33.05	20.69	64.72	42.93	30.96	22.21	144.4	65.6	38.76	31.37	129.05	115.99	31.47	30.9	138.7	70.21	34.82	24.18	152.53	60.22	38.66	32.27
Paper	5.93	5.92	5.47	4.69	9.06	8.85	4.74	4.71	8.94	8.78	5.08	4.96	8.94	8.79	5.22	4.99	8.72	8.62	4.81	4.76	14.61	9.75	4.86	4.8
$B + H_2O$	594.6	603.96	604.91	587.71	522.02	522.81	485.44	473.93	604.97	603.86	597.13	596.43	510.16	465.81	434.63	411.38	586.9	592.48	581.49	602.34	477.96	496.18	417.1	394.68
Bottle	522.02	522.81	485.44	473.93	421.68	384.38	338.21	339.22	510.16	465.81	434.63	411.38	479.3	327.14	270.08	248.71	477.96	496.18	417.1	394.68	359.97	347.47	247.67	215.02
$D + C + H_2O$	823.39	706.9	792.11	677.67	718.39	705.79	742.23	681.86	1011.09	815.39	746.65	762.52	895.93	1079.73	685.11	729.87	951.06	781.66	757.71	755.32	1020.74	762.36	782.24	825.45
Ore recovered	**77.45**	**35.58**	**27.58**	**16**	**55.66**	**34.08**	**26.22**	**17.5**	**135.46**	**56.82**	**33.68**	**26.41**	**120.11**	**107.2**	**26.25**	**25.91**	**129.98**	**61.59**	**30.01**	**19.42**	**137.92**	**50.47**	**33.8**	**27.47**
Water used	**72.58**	**81.15**	**119.47**	**113.78**	**100.34**	**138.43**	**147.23**	**134.71**	**94.81**	**138.05**	**162.5**	**185.05**	**30.86**	**138.67**	**164.55**	**162.67**	**108.94**	**96.3**	**164.39**	**207.66**	**117.99**	**148.71**	**169.43**	**179.66**
Water recovered	**314.89**	**232.56**	**277.39**	**187.11**	**203.92**	**175.67**	**201.11**	**168.87**	**422.35**	**262.91**	**182.8**	**190.28**	**386.49**	**476.25**	**126.64**	**180.51**	**353.67**	**266.16**	**195.64**	**167.46**	**406.36**	**205.57**	**211.34**	**257.54**
Cumulative ore recovered	77.45	113.03	140.61	156.61	55.66	89.74	115.96	133.46	135.46	192.28	225.96	252.37	120.11	227.31	253.56	279.47	129.98	191.57	221.58	241	137.92	188.39	222.19	249.66
Cumulative water recovered	314.89	547.45	824.84	1011.95	203.92	379.59	580.7	749.57	422.35	685.26	868.06	1058.34	386.49	862.74	989.38	1169.89	353.67	619.83	815.47	982.93	406.36	611.93	823.27	1080.81
%Cu: concentrates	7.111	4.319	1.721	1.177	6.411	4.107	2.076	1.283	6.974	1.658	0.5196	0.2692	6.997	1.827	0.6243	0.3038	10.58	0.7935	0.2928	0.1829	7.739	1.184	0.3487	0.1915
%Cu: tailings				0.62				0.6426				0.0596				0.0612				0.0426				0.0438
Cu: concentrates (g)	5.50747	1.5367	0.474652	0.18832	3.568363	1.399666	0.544327	0.224525	9.44698	0.942076	0.175001	0.071096	8.404097	1.958544	0.163879	0.078715	13.75188	0.488717	0.087869	0.035519	10.67363	0.597565	0.117861	0.052605
Cu: tailings (g)				5.229018				5.568386				0.445587				0.440964				0.323334				0.328649
Mass cumulative copper (g)	**5.50747**	**7.04417**	**7.518822**	**7.707142**	**3.568363**	**4.968028**	**5.512355**	**5.73688**	**9.44698**	**10.38906**	**10.56406**	**10.63515**	**8.404097**	**10.36264**	**10.52652**	**10.60523**	**13.75188**	**14.2406**	**14.32847**	**14.36399**	**10.67363**	**11.27119**	**11.38905**	**11.44166**
Cumulative copper recovery (%)	42.57422	54.45333	58.12252	59.57828	31.56372	43.94437	48.75918	50.7452	85.25586	93.75778	95.33711	95.97872	76.08135	93.81183	95.2954	96.008	93.63098	96.95845	97.55671	97.79855	90.68266	95.75955	96.76088	97.20781
Cumulative copper grade (%)	7.111	6.232124	5.347288	4.921232	6.411	5.536024	4.75367	4.298577	6.974	5.403087	4.675189	4.214111	6.997	4.558814	4.151491	3.794767	10.58	7.433628	6.4665	5.960161	7.739	5.982904	5.125818	4.582896
Cumulative recovered gangue (g)	61.53246	92.67113	118.8793	134.335	45.34681	75.38154	100.0283	116.8794	108.1566	162.2538	195.428	221.6326	95.8207	197.3602	223.1365	248.819	90.23467	150.4122	180.1682	199.4856	107.0714	155.8143	189.2737	216.5916
Residual concentrate SIBX (abs)									0.069	0.061	0.055	0.054	0.085	0.057	0.056	0.068	0.612	0.07	0.059	0.063	0.256	0.173	0.124	0.07
Residual tailings SIBX (abs)												0.054				0.052				0.058				0.056
Residual concentrate SIBX (ppm)									0.444995	0.393402	0.354706	0.348257	0.548183	0.367605	0.361156	0.438546	3.946915	0.451445	0.380503	0.4063	1.650997	1.115713	0.799702	0.451445
Residual tailings SIBX (ppm)												0.348257				0.335359				0.374054				0.361156

A.2. Synthetic Plant Waters at 0 g/t SIBX

Weight of parameter (g)	1SPW								3SPW								5SPW							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	61.75	38.17	24.24	19.75	60.63	40.41	25.79	19.96	93.41	55.04	26.27	18.59	105.54	48.14	28.03	19.86	120.44	67.84	29.71	19.38	106.8	67.38	27.24	17.88
Paper	6.07	4.72	4.66	4.83	6.78	5.32	4.13	4.79	6.1301	5.9446	4.8749	5.6381	6.7125	6.0366	6.9938	5.5883	4.9	4.52	4.8	4.84	5.07	4.37	4.16	3.77
B + H_2O	599.51	593.67	600.48	608.53	587.35	605.32	604.59	595.55	606.34	601.95	591.49	597.97	532.21	591.24	596.14	605.84	607.16	600.2	599.28	602.64	590.9	606.62	605.77	601.25
Bottle	417.25	289.2	186.02	95.2	387.78	328.68	215.23	114.69	532.21	414.69	359.86	322.33	450.72	424.76	361.64	368.58	500.34	437.62	351.38	259.63	473.79	452.99	324.89	202.53
D + C + H_2O	834.29	843.2	898.15	1011.48	812.16	815.63	893.55	983.75	852.17	906.34	803.4	792.67	1008.34	879.35	831.14	794.39	1132.65	1104.16	938.42	955.71	1073.67	1082.95	940.79	1003.41
Ore recovered	**55.68**	**33.45**	**19.58**	**14.92**	**53.85**	**35.09**	**21.66**	**15.17**	**87.2799**	**49.0954**	**21.3951**	**12.9519**	**98.8275**	**42.1034**	**21.0362**	**14.2717**	**115.54**	**63.32**	**24.91**	**14.54**	**101.73**	**63.01**	**23.08**	**14.11**
Water used	**182.26**	**304.47**	**414.46**	**513.33**	**199.57**	**276.64**	**389.36**	**480.86**	**74.13**	**187.26**	**231.63**	**275.64**	**81.49**	**166.48**	**234.5**	**237.26**	**106.82**	**162.58**	**247.9**	**343.01**	**117.11**	**153.63**	**280.88**	**398.72**
Water recovered	**227.29**	**145.16**	**105.67**	**108.22**	**189.68**	**143.78**	**124.09**	**112.71**	**332.2901**	**312.3746**	**182.7049**	**143.2981**	**469.5525**	**313.1566**	**207.9338**	**182.0783**	**541.23**	**518.14**	**307.17**	**223.15**	**485.77**	**506.19**	**278.39**	**215.57**
Cumulative ore recovered	55.68	89.13	108.71	123.63	53.85	88.94	110.6	125.77	87.2799	136.3753	157.7704	170.7223	98.8275	140.9309	161.9671	176.2388	115.54	178.86	203.77	218.31	101.73	164.74	187.82	201.93
Cumulative water recovered	227.29	372.45	478.12	586.34	189.68	333.46	457.55	570.26	332.2901	644.6647	827.3696	970.6677	469.5525	782.7091	990.6429	1172.721	541.23	1059.37	1366.54	1589.69	485.77	991.96	1270.35	1485.92
%Cu: concentrates	4.009	4.075	2.617	1.883	5.198	4.799	2.781	1.762	6.717	3.877	1.617	0.8839	4.413	3.842	1.932	1.233	4.516	3.086	1.916	1.319	3.772	3.124	2.259	1.629
%Cu: tailings				0.807				0.698				0.5128				0.5657				0.402				0.472
Cu: concentrates (g)	2.232211	1.363088	0.512409	0.280944	2.799123	1.683969	0.602365	0.267295	5.862591	1.903429	0.345959	0.114482	4.361258	1.617613	0.406419	0.17597	5.217786	1.954055	0.477276	0.191783	3.837256	1.968432	0.521377	0.229852
Cu: tailings (g)				7.072306				6.102125				4.252536				4.660017				3.142394				3.76689
Mass cumulative copper (g)	**2.232211**	**3.595299**	**4.107707**	**4.388651**	**2.799123**	**4.483092**	**5.085457**	**5.352752**	**5.862591**	**7.76602**	**8.111978**	**8.22646**	**4.361258**	**5.97887**	**6.38529**	**6.56126**	**5.217786**	**7.171842**	**7.649117**	**7.8409**	**3.837256**	**5.805688**	**6.327065**	**6.556917**
Cumulative copper recovery (%)	19.47666	31.36997	35.84088	38.29219	24.43608	39.13697	44.39556	46.72902	46.97967	62.23273	65.00505	65.92245	38.86597	53.28155	56.90341	58.4716	47.50657	65.29773	69.6432	71.38933	37.169	56.23592	61.28616	63.51259
Cumulative copper grade (%)	4.009	4.033769	3.778592	3.549827	5.198	5.04058	4.598062	4.255985	6.717	5.694594	5.141635	4.818621	4.413	4.242413	3.942337	3.722937	4.516	4.009752	3.753799	3.591636	3.772	3.524152	3.368686	3.247124
Cumulative recovered gangue (g)	49.22852	78.73896	96.83801	110.946	45.76005	75.98309	95.90215	110.2996	70.336	113.9302	134.3254	146.9464	86.22271	123.6509	143.5125	157.2756	100.4597	158.1321	181.6627	195.6484	90.63967	147.9606	169.5337	182.9794
Residual concentrate SIBX (abs)																								
Residual tailings SIBX (abs)																								
Residual concentrate SIBX (ppm)																								
Residual tailings SIBX (ppm)																								

A.3. 1SPW at 0.5 g/t SIBX

Weight of parameter (g)	1SPW Run 1							
	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	52.5	42.59	27.01	17.59	70.83	44.48	23.61	17.81
Paper	4.33	4.06	4.1	4.19	4.31	4.19	4.09	3.84
B + H_2O	601.61	591.44	600.48	601.24	605.53	589.74	608.12	602.33
Bottle	433.97	314.31	245.01	184.61	466.13	324.6	207.24	235.79
D + C + H_2O	747.29	841.16	877.08	922.78	808.74	871.47	909.89	882.72
Ore recovered	**48.17**	**38.53**	**22.91**	**13.4**	**66.52**	**40.29**	**19.52**	**13.97**
Water used	**167.64**	**277.13**	**355.47**	**416.63**	**139.4**	**265.14**	**400.88**	**366.54**
Water recovered	**162.42**	**165.38**	**140.26**	**117.74**	**233.76**	**205.92**	**131.05**	**127.2**
Cumulative ore recovered	48.17	86.7	109.61	123.01	66.52	106.81	126.33	140.3
Cumulative water recovered	162.42	327.8	468.06	585.8	233.76	439.68	570.73	697.93
%Cu: concentrates	6.516	4.212	2.29	1.507	8.255	3.769	1.725	1.009
%Cu: tailings				0.388				0.38
Copper: concentrates (g)	3.138757	1.622884	0.524639	0.201938	5.491226	1.51853	0.33672	0.140957
Copper: tailings (g)				3.402721				3.26686
Mass cumulative copper (g)	**3.138757**	**4.761641**	**5.28628**	**5.488218**	**5.491226**	**7.009756**	**7.346476**	**7.487433**
Cumulative copper recovery (%)	35.30288	53.55611	59.45693	61.72821	51.06078	65.181	68.31203	69.62274
Cumulative copper grade (%)	6.516	5.492089	4.822808	4.461603	8.255	6.562828	5.815306	5.336731
Cumulative recovered gangue (g)	39.09845	72.93803	94.33173	107.1481	50.6494	86.55059	105.0974	118.66

A.4. Synthetic Plant Waters at 50 g/t SIBX, Batch 1

Weight of parameter (g)	1SPW Run 1								1SPW Run 2								1SPW Run 3 (No additional frother)							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	159.43	77.18	21.98	24.74	216.64	82.79	38.9	22.31	103.69	57.6	26.59	19.36	127.79	60.33	26.64	17.32	117.66	54.25	26.72	17.19	119.28	58.63	29.22	25.23
Paper	4.29	4.28	4.09	3.99	5.89	5.04	4.46	4.4	4.57	4.35	4.07	4.06	6.28	6.91	5.62	3.33	4.66	4.57	4.65	4.15	5.59	5.53	5.01	4.86
B + H_2O	550.71	604.34	606.82	598.84	600.67	600.22	588.61	598.89	591.13	602.71	598.81	599.5	601.24	606.38	607.18	568.77	580.17	606.51	606.42	609.33	609.9	600.59	595.73	601.92
Bottle	418.93	324.94	280.24	210.48	542.92	419.38	337.01	228.98	514.27	407.35	338.84	244.1	541.08	430.96	310.25	164.02	511.42	416.56	302.89	279.63	524.67	529.43	504.41	540.69
D + C + H_2O	1121.66	1083.67	819.25	961.27	1346.99	1051.15	960.15	999.09	991.45	981.24	863.19	936.39	1154.94	1046.89	934.86	989.79	1115.29	1001.24	928.46	896.65	1000.95	995.62	975.68	810.32
Ore recovered	**155.14**	**72.9**	**17.89**	**20.75**	**210.75**	**77.75**	**34.44**	**17.91**	**99.12**	**53.25**	**22.52**	**15.3**	**121.51**	**53.42**	**21.02**	**13.99**	**113**	**49.68**	**22.07**	**13.04**	**113.69**	**53.1**	**24.21**	**20.37**
Water used	**131.78**	**279.4**	**326.58**	**388.36**	**57.75**	**180.84**	**251.6**	**369.91**	**76.86**	**195.36**	**259.97**	**355.4**	**60.16**	**175.42**	**296.93**	**404.75**	**68.75**	**189.95**	**303.53**	**329.7**	**85.23**	**71.16**	**91.32**	**61.23**
Water recovered	**465.68**	**371.25**	**116.34**	**177.15**	**709.43**	**432.44**	**315.67**	**236.26**	**446.41**	**372.51**	**222.26**	**190.68**	**604.21**	**457.93**	**258.47**	**196.04**	**564.48**	**401.49**	**244.42**	**178.9**	**432.97**	**511.24**	**501.71**	**353.71**
Cumulative ore recovered	155.14	228.04	245.93	266.68	210.75	288.5	322.94	340.85	99.12	152.37	174.89	190.19	121.51	174.93	195.95	209.94	113	162.68	184.75	197.79	113.69	166.79	191	211.37
Cumulative water recovered	465.68	836.93	953.27	1130.42	709.43	1141.87	1457.54	1693.8	446.41	818.92	1041.18	1231.86	604.21	1062.14	1320.61	1516.65	564.48	965.97	1210.39	1389.29	432.97	944.21	1445.92	1799.63
%Cu: concentrates	6.661	0.886	0.41	0.206	5.506	0.991	2.466	0.255	3.09	3.196	2.329	1.846	4.093	3.324	0.333	1.954	2.41	2.907	2.481	2.136	3.805	1.726	1.02	0.783
%Cu: tailings				0.0929				0.102				0.513				0.422				0.518				0.296
Copper: concentrates (g)	10.33388	0.645894	0.073349	0.042745	11.6039	0.770503	0.84929	0.045671	3.062808	1.70187	0.524491	0.282438	4.973404	1.775681	0.069997	0.273365	2.7233	1.444198	0.547557	0.278534	4.325905	0.916506	0.246942	0.159497
Copper: tailings (g)				0.681254				0.672333				4.154325				3.334053				4.155448				2.334345
Mass cumulative copper (g)	**10.33388**	**10.97977**	**11.05312**	**11.09586**	**11.6039**	**12.3744**	**13.22369**	**13.26936**	**3.062808**	**4.764678**	**5.289169**	**5.571607**	**4.973404**	**6.749085**	**6.819082**	**7.092446**	**2.7233**	**4.167498**	**4.715054**	**4.993589**	**4.325905**	**5.242411**	**5.489353**	**5.64885**
Cumulative copper recovery (%)	87.74537	93.22968	93.85249	94.21544	83.23162	88.75822	94.84995	95.17754	31.49115	48.98942	54.38213	57.2861	47.69966	64.73011	65.40145	68.02327	29.76598	45.55122	51.53607	54.58049	54.18764	65.66808	68.76135	70.75926
Cumulative copper grade (%)	6.661	4.814844	4.494416	4.160741	5.506	4.289219	4.094782	3.89302	3.09	3.127045	3.024283	2.929495	4.093	3.858163	3.480011	3.378321	2.41	2.561776	2.552127	2.524692	3.805	3.14312	2.874007	2.672494
Cumulative recovered gangue (g)	125.2733	196.3066	213.9846	234.611	177.2127	252.7358	284.7212	302.4993	90.26795	138.5993	159.6034	174.0871	107.136	155.424	176.2417	189.4416	105.1292	150.6352	171.1227	183.3577	101.1874	151.6385	175.1348	195.0438
Residual concentrate SIBX (abs)	0.081	0.081	0.078	0.075	0.085	0.085	0.081	0.074																
Residual tailings SIBX (abs)				0.075				0.072																
Residual concentrate SIBX (mol/l)	6.15E-06	6.15E-06	5.92E-06	5.7E-06	6.46E-06	6.46E-06	6.15E-06	5.62E-06																
Residual tailings SIBX (mol/l)				5.7E-06				5.47E-06																

Weight of parameter (g)	3SPW Run 1								3SPW Run 2								3SPW Run 3 (No additional frother)							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	99.68	66.93	39.97	22.96	121.78	64.27	36.82	23.28	95.87	57.03	25.76	18.73	104.28	52.91	29.84	20.64	38.28	30.45	23.94	20.35	53.16	36.59	19.57	16.65
Paper	6.37	6.15	5.78	5.93	6.42	5.79	5.7	5.61	6.19	5.95	5.94	5.57	5.95	5.78	5.62	4.98	6.24	6.01	5.47	5.71	5.72	5.25	5.56	4.83
B + H_2O	599.07	584.63	599.95	584.66	595.81	598.19	605.48	591.79	588.05	591.93	595.47	598.03	608.89	600.23	598.41	599.72	600.75	605.18	603.28	602.29	598.54	596.32	605.17	603.38
Bottle	478.9	426.5	398.23	297.36	446.6	374.9	346.64	251.47	475.68	425.28	378.44	272.51	487.64	430.69	385.58	295.87	450.68	377.9	363.26	296.27	478.44	429.75	392.58	297.13
D + C + H_2O	826.61	870.86	869.56	846.94	956.21	936.94	900.01	905.68	1010.41	997.84	817.85	871.38	1000.32	995.64	800.29	872.38	669.14	724.14	748.09	817.87	744.72	755.97	703.55	800.62
Ore recovered	**93.31**	**60.78**	**34.19**	**17.03**	**115.36**	**58.48**	**31.12**	**17.67**	**89.68**	**51.08**	**19.82**	**13.16**	**98.33**	**47.13**	**24.22**	**15.66**	**32.04**	**24.44**	**18.47**	**14.64**	**47.44**	**31.34**	**14.01**	**11.82**
Water used	**120.17**	**158.13**	**201.72**	**287.3**	**149.21**	**223.29**	**258.84**	**340.32**	**112.37**	**166.65**	**217.03**	**325.52**	**121.25**	**169.54**	**212.83**	**303.85**	**150.07**	**227.28**	**240.02**	**306.02**	**120.1**	**166.57**	**212.59**	**306.25**
Water recovered	**254.66**	**294.34**	**265.98**	**181.83**	**333.17**	**297.56**	**242.38**	**186.91**	**449.89**	**422.5**	**213.33**	**171.92**	**422.27**	**421.36**	**195.57**	**192.09**	**128.56**	**114.81**	**121.93**	**136.43**	**218.71**	**200.45**	**109.28**	**121.77**
Cumulative ore recovered	93.31	154.09	188.28	205.31	115.36	173.84	204.96	222.63	89.68	140.76	160.58	173.74	98.33	145.46	169.68	185.34	32.04	56.48	74.95	89.59	47.44	78.78	92.79	104.61
Cumulative water recovered	254.66	549	814.98	996.81	333.17	630.73	873.11	1060.02	449.89	872.39	1085.72	1257.64	422.27	843.63	1039.2	1231.29	128.56	243.37	365.3	501.73	218.71	419.16	528.44	650.21
%Cu: concentrates	10.05	0.951	0.2538	0.2242	9.284	9.6862	0.2804	0.2076	5.2	4.358	2.735	1.59	4.42	4.39	2.331	1.605	2.324	2.564	2.311	1.946	2.447	2.554	2.516	2.118
%Cu: tailings				0.0712				0.0749				0.3634				0.3436				1.194				1.072
Copper: concentrates (g)	9.377655	0.578018	0.086774	0.038181	10.71002	5.66449	0.08726	0.036683	4.66336	2.226066	0.542077	0.209244	4.346186	2.069007	0.564568	0.251343	0.74461	0.626642	0.426842	0.284894	1.160857	0.800424	0.352492	0.250348
Copper: tailings (g)				0.565819				0.58225				3.002629				2.799172				10.8703				9.598581
Mass cumulative copper (g)	**9.377655**	**9.955673**	**10.04245**	**10.08063**	**10.71002**	**16.37451**	**16.46177**	**16.49846**	**4.66336**	**6.889426**	**7.431503**	**7.640747**	**4.346186**	**6.415193**	**6.979761**	**7.231104**	**0.74461**	**1.371251**	**1.798093**	**2.082987**	**1.160857**	**1.96128**	**2.313772**	**2.56412**
Cumulative copper recovery (%)	88.08248	93.51169	94.32674	94.68537	62.70246	95.86555	96.37642	96.59118	43.81467	64.72971	69.8228	71.78876	43.33067	63.95829	69.58693	72.09277	5.748424	10.58613	13.88137	16.08077	9.5444	16.12537	19.02351	21.08183
Cumulative copper grade (%)	10.05	6.460947	5.333783	4.909955	9.284	9.419301	8.0317	7.410706	5.2	4.894449	4.627913	4.397806	4.42	4.41028	4.113485	3.901535	2.324	2.427853	2.399057	2.325022	2.447	2.489566	2.493557	2.451123
Cumulative recovered gangue (g)	66.20695	125.3164	159.2556	176.1752	84.40618	126.5148	157.3826	174.9466	76.20208	120.8484	139.1017	151.6569	85.76877	126.919	149.5073	164.4409	29.88795	52.51685	69.7532	83.56981	44.08492	73.11156	86.1028	97.19925
Residual concentrate SIBX (abs)	0.136848	0.133643	0.132606	0.132	0.138	0.137717	0.134	0.132																
Residual tailings SIBX (abs)				0.127				0.13																
Residual concentrate SIBX (ppm)	8.97E-06	8.76E-06	8.69E-06	8.65E-06	9.05E-06	9.03E-06	8.78E-06	8.65E-06																
Residual tailings SIBX (ppm)				8.32E-06				8.52E-06																

Weight of parameter (g)	5SPW Run 1				5SPW Run 2								5SPW Run 3 (No additional frother dosed)							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	195.97	88.12	34.99	24.26	260.82	86.44	36.9	22.58	94.23	65.5	32.65	18.75	133.1	64.67	33.29	18.79	74.64	39.14	24.13	17.07	68.04	32.83	18.84	17.79
Paper	5.28	5	4.96	4.18	5.03	4.64	4.56	4.05	5.13	5.09	4.99	3.6	5.84	5.64	4.53	4.13	6.07	5.79	5.68	3.98	5.59	4.61	4.58	4.32
B + H2O	593.45	594.24	604	601.3	601.3	596.26	607.86	595.84	600.49	606.92	604.26	597.75	590.4	612.09	605.58	605.82	611.11	603.46	601.65	608.17	488.04	606.55	596.16	600.06
Bottle	539.56	433.66	304.35	208.67	511.2	410.06	356.83	249.31	511.07	496.04	366.27	268.89	488.19	421.25	298.82	262.25	480.81	391.31	288.31	224.36	347.46	391.62	229.79	171.13
D + C + H2O	1267.16	1246.75	1089.37	1122.91	1632.11	1213.22	1105.49	1045.47	1032.12	1099.14	1014.88	986.05	1266.65	1148.62	1127.48	960.89	879.01	826.37	834.19	901.91	879.07	784.64	877.59	910.66
Ore recovered	**190.69**	**83.12**	**30.03**	**20.08**	**255.79**	**81.8**	**32.34**	**18.53**	**89.1**	**60.41**	**27.66**	**15.15**	**127.26**	**59.03**	**28.76**	**14.66**	**68.57**	**33.35**	**18.45**	**13.09**	**62.45**	**28.22**	**14.26**	**13.47**
Water used	**53.89**	**160.58**	**299.65**	**392.63**	**90.1**	**186.2**	**251.03**	**346.53**	**89.42**	**110.88**	**237.99**	**328.86**	**102.21**	**190.84**	**306.76**	**343.57**	**130.3**	**212.15**	**313.34**	**383.81**	**140.58**	**214.93**	**366.37**	**428.93**
Water recovered	**653.52**	**642.93**	**401.25**	**335.19**	**917.16**	**585.1**	**463.68**	**305.4**	**484.54**	**567.73**	**390.79**	**267.03**	**668.12**	**538.63**	**433.52**	**227.65**	**311.08**	**220.75**	**143.96**	**130**	**306.98**	**181.37**	**138.52**	**93.25**
Cumulative ore recovered	190.69	273.81	303.84	323.92	255.79	337.59	369.93	388.46	89.1	149.51	177.17	192.32	127.26	186.29	215.05	229.71	68.57	101.92	120.37	133.46	62.45	90.67	104.93	118.4
Cumulative water recovered	653.52	1296.45	1697.7	2032.89	917.16	1502.26	1965.94	2271.34	484.54	1052.27	1443.06	1710.09	668.12	1206.75	1640.27	1867.92	311.08	531.83	675.79	805.79	306.98	488.35	626.87	720.12
%Cu: concentrates	5.485	0.475	0.233	2.288	3.707	0.472	0.235	0.179	2.447	3.073	2.874	2.353	3.033	3.201	2.131	1.711	3.085	3.632	2.877	0.176	2.518	3.103	2.834	2.16
%Cu: tailings				0.085				0.197				0.475				0.375				0.563				0.599
Copper: concentrates (g)	10.45935	0.39482	0.06997	0.45943	9.482135	0.386096	0.075999	0.033169	2.180277	1.856399	0.794948	0.35648	3.859796	1.88955	0.612876	0.250833	2.115385	1.211272	0.530807	0.023038	1.572491	0.875667	0.404128	0.290952
Copper: tailings (g)				0.574668				1.204734				3.83648				2.888588				4.87862				5.280784
Mass cumulative copper (g)	**10.45935**	**10.85417**	**10.92414**	**11.38357**	**9.482135**	**9.868231**	**9.94423**	**9.977399**	**2.180277**	**4.036676**	**4.831625**	**5.188104**	**3.859796**	**5.749346**	**6.362222**	**6.613054**	**2.115385**	**3.326657**	**3.857463**	**3.880501**	**1.572491**	**2.448158**	**2.852286**	**3.143238**
Cumulative copper recovery (%)	87.46564	90.7673	91.35242	95.19437	84.7972	88.24999	88.92964	89.22626	24.15931	44.72978	53.53847	57.48857	40.62241	60.50898	66.95918	69.59907	24.15065	37.97934	44.03938	44.3024	18.66675	29.06162	33.85896	37.31279
Cumulative copper grade (%)	5.485	3.964123	3.595358	3.514314	3.707	2.923141	2.688138	2.56845	2.447	2.699937	2.727112	2.697642	3.033	3.086234	2.958485	2.878871	3.085	3.263988	3.204671	2.907614	2.518	2.700075	2.718275	2.654762
Cumulative recovered gangue (g)	160.4607	242.4396	272.2674	291.0195	228.385	309.0691	341.1895	359.6236	82.79862	137.8433	163.2058	177.3255	116.1045	169.6734	196.6621	210.5971	62.45617	92.30539	109.2213	122.2447	57.90523	83.5944	96.6864	109.3155
Residual concentrate SIBX (abs)	0.284	0.171	0.17	0.159	0.187	0.18	0.176	0.17																
Residual tailings SIBX (abs)				0.157				0.169																
Residual concentrate SIBX (ppm)	1.84E-05	1.11E-05	1.1E-05	1.03E-05	1.21E-05	1.17E-05	1.14E-05	1.1E-05																
Residual tailings SIBX (ppm)				1.02E-05				1.1E-05																

A.5. Synthetic Plant Waters at 100 g/t SIBX, Batch 1

Weight of parameter (g)	1SPW Run 1				1SPW Run 2								1SPW Run 3 (No additional frother dosed)							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	161.47	42.21	24.88	21.68	161.43	42.21	28.45	22.11	119.2	73.83	30.5	18.26	105.3	59.58	36.05	21.7	46.92	30.49	19.19	17.91	31.38	27.87	19.72	16.52
Paper	6.43	5.99	5.84	5.79	7.18	6.94	6.64	6.43	5.75	5.6	5.24	4.94	6.65	6.59	6.14	5.61	5.68	5.48	5.47	5.44	5.62	6.6	5.6	5.75
B + H_2O	608.56	600.96	600.66	598.68	596.18	597.65	609.99	585.13	604.38	598.09	609.04	604.99	597.85	591.34	599.21	590.71	604.88	605.36	598.87	609.66	600.87	598.46	595.28	595.52
Bottle	466.18	398.78	314.1	234.79	448.21	367	309.54	263.77	499.89	446.54	363.01	255.29	479.72	410.5	307.68	250.9	480.53	283.14	381.78	250.2	437.05	327.8	251.97	182.17
D + C + H_2O	1158.3	813.59	821.5	887.19	1107.68	788.81	833	828.89	1068.98	1146.95	967.01	927.64	1056.86	1022.39	1078.74	979.85	590.66	721.41	773.62	809.41	638.42	733.63	798.85	835.36
Ore recovered	**155.04**	**36.22**	**19.04**	**15.89**	**154.25**	**35.27**	**21.81**	**15.68**	**113.45**	**68.23**	**25.26**	**13.32**	**98.65**	**52.99**	**29.91**	**16.09**	**41.24**	**25.01**	**13.72**	**12.47**	**25.76**	**21.27**	**14.12**	**10.77**
Water used	**142.38**	**202.18**	**286.56**	**363.89**	**147.97**	**230.65**	**300.45**	**321.36**	**104.49**	**151.55**	**246.03**	**349.7**	**118.13**	**180.84**	**291.53**	**339.81**	**124.35**	**322.22**	**217.09**	**359.46**	**163.82**	**270.66**	**343.31**	**413.35**
Water recovered	**490.96**	**215.08**	**141.38**	**146.46**	**435.54**	**162.78**	**136.22**	**130.9**	**481.12**	**567.06**	**321.2**	**203.67**	**470.16**	**428.45**	**382.78**	**263**	**55.15**	**14.07**	**168.29**	**76.53**	**78.92**	**81.59**	**66.9**	**50.29**
Cumulative ore recovered	155.04	191.26	210.3	226.19	154.25	189.52	211.33	227.01	113.45	181.68	206.94	220.26	98.65	151.64	181.55	197.64	41.24	66.25	79.97	92.44	25.76	47.03	61.15	71.92
Cumulative water recovered	490.96	706.04	847.42	993.88	435.54	598.32	734.54	865.44	481.12	1048.18	1369.38	1573.05	470.16	898.61	1281.39	1544.39	55.15	69.22	237.51	314.04	78.92	160.51	227.41	277.7
%Cu: concentrates	8.04	0.49	0.22	0.15	7.24	0.62	0.22	0.16	9.24	2.20	1.11	0.86	4.38	3.81	2.06	1.38	3.96	4.16	4.02	2.51	1.79	1.86	1.95	2.07
%Cu: tailings				0.06				0.06				0.22				0.27				0.86				1.01
Copper: concentrates (g)	12.46072	0.175703	0.041469	0.0242	11.17248	0.219344	0.048942	0.025856	10.48062	1.503857	0.281169	0.114858	4.315938	2.01627	0.61498	0.221575	1.633764	1.039191	0.551695	0.313134	0.461645	0.396494	0.275778	0.223381
Copper: tailings (g)				0.464286				0.463794				1.715428				2.166372				7.805016				9.373608
Mass cumulative copper (g)	**12.46072**	**12.63642**	**12.67789**	**12.70209**	**11.17248**	**11.39183**	**11.44077**	**11.46662**	**10.48062**	**11.98448**	**12.26565**	**12.38051**	**4.315938**	**6.332207**	**6.947187**	**7.168762**	**1.633764**	**2.672954**	**3.224649**	**3.537783**	**0.461645**	**0.858139**	**1.133917**	**1.357297**
Cumulative copper recovery (%)	94.64045	95.97493	96.2899	96.4737	93.64703	95.48556	95.89578	96.11251	74.35209	85.02082	87.0155	87.83034	46.23327	67.83199	74.41978	76.79335	14.40353	23.56521	28.42904	31.18969	4.302013	7.996893	10.56683	12.64849
Cumulative copper grade (%)	8.0371	6.606935	6.028479	5.615674	7.2431	6.010883	5.413698	5.051154	9.2381	6.596478	5.927153	5.620861	4.375	4.175816	3.826597	3.627182	3.9616	4.034648	4.032324	3.827113	1.7921	1.824663	1.85432	1.887232
Cumulative recovered gangue (g)	119.0264	154.7385	173.6587	189.4787	121.9596	156.5956	178.2642	193.8695	83.15918	147.0428	171.4901	184.4782	86.17619	133.3388	161.4714	176.921	36.51814	58.5247	70.6502	82.21519	24.42577	44.54983	57.87278	67.99718
Residual concentrate SIBX (abs)	0.279	0.084	0.083	0.082	0.292	0.089	0.084	0.081																
Residual tailings SIBX (abs)				0.077				0.078																
Residual concentrate SIBX (ppm)	1.94E-05	5.85E-06	5.78E-06	5.71E-06	2.03E-05	6.2E-06	5.85E-06	5.64E-06																
Residual tailings SIBX (ppm)				5.36E-06				5.43E-06																

Weight of parameter (g)	3SPW Run 1				3SPW Run 2								3SPW Run 3 (No additional frother dosed)							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	160.65	59.04	39.73	27.49	179.79	47.39	34.91	31.93	130.41	75.16	30.14	22.01	135.34	73.84	30.14	24.77	50.28	40.6	23.85	28.88	45.36	28.27	32.41	34.09
Paper	7.3	6.2	4.89	4.82	6.1	5.83	4.62	4.55	6.07	5.2	4.91	4.88	5.92	4.61	4.56	4.41	6.35	5.75	5.18	4.75	4.76	4.59	4.5	4.31
B + H_2O	605.22	606.27	601.32	603.06	604.41	603.11	606.53	598.57	610.3	607.68	592.41	602.88	585.51	598.42	602.3	601.74	607.03	606.78	607.43	601.59	489.75	610.01	598.83	600.35
Bottle	491.79	497.16	414.84	414.32	506.43	479.2	445.35	370.52	539.85	490.68	489.04	352.01	489.75	535.1	473.5	434.91	446.63	483.89	417.4	376.37	349.63	458.87	370.68	358.87
D + C + H_2O	1101.39	845.25	916.98	878.29	1156.21	767.32	827.88	945.73	1126.55	1180.09	851.99	933.38	1234.91	1145.01	889.7	918.5	764.26	756.05	745.9	861.9	734.32	675.93	843.76	902.76
Ore recovered	**153.35**	**52.84**	**34.84**	**22.67**	**173.69**	**41.56**	**30.29**	**27.38**	**124.34**	**69.96**	**25.23**	**17.13**	**129.42**	**69.23**	**25.58**	**20.36**	**43.93**	**34.85**	**18.67**	**24.13**	**40.6**	**23.68**	**27.91**	**29.78**
Water used	**113.43**	**109.11**	**186.48**	**188.74**	**97.98**	**123.91**	**161.18**	**228.05**	**70.45**	**117**	**103.37**	**250.87**	**95.76**	**63.32**	**128.8**	**166.83**	**160.4**	**122.89**	**190.03**	**225.22**	**140.12**	**151.14**	**228.15**	**241.48**
Water recovered	**476.14**	**325.69**	**327.99**	**306.1**	**526.07**	**244.24**	**268.74**	**329.52**	**573.29**	**635.52**	**355.72**	**304.6**	**651.26**	**654.85**	**367.65**	**370.53**	**201.46**	**240.7**	**169.53**	**251.77**	**195.13**	**143.5**	**220.03**	**270.72**
Cumulative ore recovered	153.35	206.19	241.03	263.7	173.69	215.25	245.54	272.92	124.34	194.3	219.53	236.66	129.42	198.65	224.23	244.59	43.93	78.78	97.45	121.58	40.6	64.28	92.19	121.97
Cumulative water recovered	476.14	801.83	1129.82	1435.92	526.07	770.31	1039.05	1368.57	573.29	1208.81	1564.53	1869.13	651.26	1306.11	1673.76	2044.29	201.46	442.16	611.69	863.46	195.13	338.63	558.66	829.38
%Cu: concentrates	6.705	0.4423	0.1824	0.1534	8.383	0.6553	0.2538	0.1681	4.401	3.486	2.107	1.281	3.448	2.604	2.001	1.144	1.981	2.352	1.917	1.227	2.465	2.968	1.93	1.489
%Cu: tailings				0.0549				0.0716				0.4529				0.3659				1.12				1.234
Copper: concentrates (g)	10.28212	0.233711	0.063548	0.034776	14.56043	0.272343	0.076876	0.046026	5.472203	2.438806	0.531596	0.219435	4.462402	1.802749	0.511856	0.232918	0.870253	0.819672	0.357904	0.296075	1.00079	0.702822	0.538663	0.443424
Copper: tailings (g)				0.404229				0.520589				3.457167				2.764045				9.838304				10.83489
Mass cumulative copper (g)	**10.28212**	**10.51583**	**10.57938**	**10.61415**	**14.56043**	**14.83278**	**14.90965**	**14.95568**	**5.472203**	**7.911009**	**8.442605**	**8.66204**	**4.462402**	**6.265151**	**6.777007**	**7.009925**	**0.870253**	**1.689925**	**2.047829**	**2.343904**	**1.00079**	**1.703612**	**2.242275**	**2.6857**
Cumulative copper recovery (%)	93.31786	95.43896	96.01571	96.33132	94.08233	95.84208	96.33881	96.63621	45.15315	65.27662	69.66301	71.47366	45.65598	64.10037	69.3373	71.72034	7.143642	13.87208	16.81	19.24039	7.40197	12.60013	16.58415	19.86378
Cumulative copper grade (%)	6.705	5.100067	4.389237	4.025086	8.383	6.890953	6.072188	5.479876	4.401	4.071543	3.845764	3.66012	3.448	3.153864	3.022346	2.86599	1.981	2.14512	2.101415	1.92787	2.465	2.650299	2.432233	2.201935
Cumulative recovered gangue (g)	123.6329	175.7974	210.4538	233.0233	131.6078	172.3807	202.4485	229.6955	108.5244	171.4358	195.1294	211.6252	116.5229	180.5426	204.6433	224.3301	41.41482	73.89582	91.53142	114.8057	37.70754	59.35626	85.70944	114.2079
Residual concentrate SIBX (abs)	0.409	0.127	0.124	0.124	0.42	0.132	0.127	0.12																
Residual tailings SIBX (abs)				0.121				0.117																
Residual concentrate SIBX (ppm)	2.68E-05	8.32E-06	8.13E-06	8.13E-06	2.75E-05	8.65E-06	8.32E-06	7.87E-06																
Residual tailings SIBX (ppm)				7.93E-06				7.67E-06																

Weight of parameter (g)	5SPW Run 1				5SPW Run 2								5SPW Run 3 (No additional frother dosed)							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	150.66	69.29	32.32	17.79	165.44	50.43	27.63	20.57	116.01	85.88	44.12	20.99	117.01	75.08	47.56	23.86	57.54	35.82	26.85	19.9	45.56	29.4	23.41	17.53
Paper	5.68	5.66	5.39	5.07	5.29	5.7	5.36	5.97	5.86	5.59	5.53	5.34	6.05	5.69	5.62	5.29	5.89	5.76	5.43	5.13	6.01	6.13	5.71	5.37
B + H2O	608.21	591.29	602.57	595.02	507.4	535.4	602.66	577.28	608.46	601.91	603.25	606.95	582.2	599.79	576.13	606.99	608.46	596.63	612.95	595.84	499.21	478.67	485.45	591.46
Bottle	550.01	384.53	312.6	207.52	357.49	304.01	316.84	229.31	516.28	464.56	389.18	312.72	464.71	435.05	358.46	332.53	474.46	361.36	241.17	204.38	334.37	181.52	182.16	219.56
D + C + H2O	1046.45	1106.45	1004.69	942.12	1208.42	960.73	920.02	918.38	1136.09	1293.46	1198.59	984.19	1186.24	1228.92	1256.17	1032.29	772.61	774.63	912.17	898.05	730.14	769.66	784.13	821.38
Ore recovered	**144.98**	**63.63**	**26.93**	**12.72**	**160.15**	**44.73**	**22.27**	**14.6**	**110.15**	**80.29**	**38.59**	**15.65**	**110.96**	**69.39**	**41.94**	**18.57**	**51.65**	**30.06**	**21.42**	**14.77**	**39.55**	**23.27**	**17.7**	**12.16**
Water used	**58.2**	**206.76**	**289.97**	**387.5**	**149.91**	**231.39**	**285.82**	**347.97**	**92.18**	**137.35**	**214.07**	**294.23**	**117.49**	**164.74**	**217.67**	**274.46**	**134**	**235.27**	**371.78**	**391.46**	**164.84**	**297.15**	**303.29**	**371.9**
Water recovered	**473.35**	**475.95**	**313.27**	**180.95**	**528.44**	**324.5**	**237.41**	**194.86**	**563.84**	**715.71**	**571.41**	**313.36**	**587.87**	**634.68**	**622.04**	**378.31**	**217.04**	**149.19**	**144.45**	**130.87**	**155.83**	**89.13**	**88.62**	**76.37**
Cumulative ore recovered	144.98	208.61	235.54	248.26	160.15	204.88	227.15	241.75	110.15	190.44	229.03	244.68	110.96	180.35	222.29	240.86	51.65	81.71	103.13	117.9	39.55	62.82	80.52	92.68
Cumulative water recovered	473.35	949.3	1262.57	1443.52	528.44	852.94	1090.35	1285.21	563.84	1279.55	1850.96	2164.32	587.87	1222.55	1844.59	2222.9	217.04	366.23	510.68	641.55	155.83	244.96	333.58	409.95
%Cu: concentrates	8.92	0.32	0.17	0.12	7.36	0.40	0.20	0.14	3.77	3.14	1.94	1.78	2.17	2.42	2.05	1.96	2.09	3.10	2.30	1.95	4.17	4.84	4.05	2.55
%Cu: tailings				0.05				0.06				0.29				0.36				0.71				0.86
Copper: concentrates (g)	12.92801	0.202407	0.044731	0.015798	11.7904	0.177824	0.044985	0.020542	4.153316	2.517172	0.749842	0.278116	2.404614	1.680001	0.861741	0.364733	1.076954	0.93189	0.492467	0.28803	1.64967	1.125593	0.71616	0.309484
Copper: tailings (g)				0.37587				0.45495				2.190428				2.732904				6.26291				7.802952
Mass cumulative copper (g)	**12.92801**	**13.13042**	**13.17515**	**13.19095**	**11.7904**	**11.96823**	**12.01321**	**12.03375**	**4.153316**	**6.670488**	**7.42033**	**7.698446**	**2.404614**	**4.084615**	**4.946357**	**5.31109**	**1.076954**	**2.008844**	**2.501311**	**2.789341**	**1.64967**	**2.775263**	**3.491423**	**3.800907**
Cumulative copper recovery (%)	95.29141	96.78334	97.11304	97.22949	94.40853	95.83241	96.19262	96.35711	41.99989	67.45447	75.03716	77.84957	29.89329	50.77845	61.4913	66.02553	11.89709	22.19165	27.63193	30.81378	14.21656	23.91673	30.08846	32.75554
Cumulative copper grade (%)	8.9171	6.294242	5.593593	5.31336	7.3621	5.841579	5.288669	4.977768	3.7706	3.502672	3.239894	3.146332	2.1671	2.264827	2.225182	2.205053	2.0851	2.458505	2.425397	2.365853	4.1711	4.417802	4.336094	4.101108
Cumulative recovered gangue (g)	107.6158	170.6608	197.4615	210.1359	126.0737	170.2897	192.4297	206.9704	98.1462	171.1611	207.584	222.4302	104.0102	168.5448	207.9942	225.51	48.53742	75.90409	95.90078	109.8383	34.78217	54.79901	70.42918	81.69472
Residual concentrate SIBX (abs)	0.52	0.164	0.165	0.164	0.532	0.168	0.16	0.159																
Residual tailings SIBX (abs)				0.155				0.152																
Residual concentrate SIBX (ppm)	3.37E-05	1.06E-05	1.07E-05	1.06E-05	3.45E-05	1.09E-05	1.04E-05	1.03E-05																
Residual tailings SIBX (ppm)				1.01E-05				9.87E-06																

A.6. Synthetic Plant Waters at 50 g/t SIBX, Batch 2

Weight of parameter (g)	1SPW Run 1								1SPW Run 2								1SPW Run 3							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	153.71	77.35	30.89	23.41	142.34	62.25	25.15	22.7	131.62	72.03	31.21	31.57	124.91	71.56	33.41	29.25	129.47	75.13	32.47	29.5	130.49	34.42	68.31	29.43
Paper	5.56	4.73	4.67	4.61	4.91	4.85	4.74	4.55	5.04	4.86	4.85	4.55	4.94	4.85	4.64	4.38	5.13	5.14	5.06	4.96	4.65	4.62	4.62	4.37
B + H_2O	605.11	605.76	598.25	602.97	600.96	597.05	598.45	602.39	601.08	608.92	600.25	600.97	593.91	601.82	597.95	604.96	599.61	596.69	596.25	591.8	598.11	604.27	606.1	591.66
Bottle	574.75	444.79	388.45	430.28	456.29	460.61	336.6	312.81	427.8	408.87	391.97	406.19	421.83	445.8	294.58	222.76	457.45	368.33	297.23	390.33	486.93	467.1	273.84	297.39
D + C + H_2O	1050.93	966.84	780.31	732.69	1056.2	875.77	810.32	854.48	1066.81	955.49	795.16	865.78	1014.3	889.12	904.67	1014.02	997.97	969.86	873.68	815.72	987.05	888.32	955.63	956.01
Ore recovered	**148.15**	**72.62**	**26.22**	**18.8**	**137.43**	**57.4**	**20.41**	**18.15**	**126.58**	**67.17**	**26.36**	**27.02**	**119.97**	**66.71**	**28.77**	**24.87**	**124.34**	**69.99**	**27.41**	**24.54**	**125.84**	**29.8**	**63.69**	**25.06**
Water used	**30.36**	**160.97**	**209.8**	**172.69**	**144.67**	**136.44**	**261.85**	**289.58**	**173.28**	**200.05**	**208.28**	**194.78**	**172.08**	**156.02**	**303.37**	**382.2**	**142.16**	**228.36**	**299.02**	**201.47**	**111.18**	**137.17**	**332.26**	**294.27**
Water recovered	**513.95**	**375.64**	**176.62**	**180.42**	**415.63**	**324.32**	**160.39**	**185.97**	**408.48**	**330.66**	**192.85**	**283.2**	**363.78**	**308.78**	**204.86**	**246.17**	**373**	**313.9**	**179.58**	**228.93**	**391.56**	**363.74**	**192.01**	**275.9**
Cumulative ore recovered	148.15	220.77	246.99	265.79	137.43	194.83	215.24	233.39	126.58	193.75	220.11	247.13	119.97	186.68	215.45	240.32	124.34	194.33	221.74	246.28	125.84	155.64	219.33	244.39
Cumulative water recovered	513.95	889.59	1066.21	1246.63	415.63	739.95	900.34	1086.31	408.48	739.14	931.99	1215.19	363.78	672.56	877.42	1123.59	373	686.9	866.48	1095.41	391.56	755.3	947.31	1223.21
%Cu: concentrates	6.466	1.162	0.3601	0.2211	7.725	0.5002	0.2721	0.1975	8.516	0.8365	0.3062	0.1722	8.708	0.9185	0.3202	0.1905	9.903	1.214	0.3015	0.1854	7.889	0.755	0.2575	0.2027
%Cu: tailings				0.0834				0.0691				0.0669				0.0841				0.0639				0.0735
Copper: concentrates (g)	9.579379	0.843844	0.094418	0.041567	10.61647	0.287115	0.055536	0.035846	10.77955	0.561877	0.080714	0.046528	10.44699	0.612731	0.092122	0.047377	12.31339	0.849679	0.082641	0.045497	9.927518	0.22499	0.164002	0.050797
Copper: tailings (g)				0.612331				0.529728				0.50367				0.638891				0.481627				0.555373
Mass cumulative copper (g)	**9.579379**	**10.42322**	**10.51764**	**10.55921**	**10.61647**	**10.90358**	**10.95912**	**10.99496**	**10.77955**	**11.34143**	**11.42214**	**11.46867**	**10.44699**	**11.05972**	**11.15184**	**11.19922**	**12.31339**	**13.16307**	**13.24571**	**13.29121**	**9.927518**	**10.15251**	**10.31651**	**10.36731**
Cumulative copper recovery (%)	85.748065	93.30158	94.14675	94.51883	92.11932	94.61062	95.0925	95.40354	90.03712	94.73025	95.40442	95.79305	88.24879	93.42471	94.20289	94.6031	89.40346	95.57269	96.17272	96.50306	90.88903	92.94888	94.45035	94.91541
Cumulative copper grade (%)	6.466	4.721304	4.258327	3.972764	7.725	5.59646	5.091581	4.710983	8.516	5.853641	5.189289	4.640745	8.708	5.924426	5.176069	4.660127	9.903	6.773565	5.973532	5.396787	7.889	6.523071	4.703647	4.242115
Cumulative recovered gangue (g)	120.46393	190.6451	216.5922	235.2721	106.7466	163.3168	183.5662	201.6126	95.42522	160.9713	187.098	213.9835	89.77639	154.7155	183.2192	207.9523	88.75217	156.2864	183.4576	207.8661	97.14775	126.2975	189.5135	214.4267
Residual concentrate SIBX (abs)	0.081	0.081	0.078	0.075	0.085	0.085	0.081	0.074	0.088	0.084	0.084	0.079	0.292	0.089	0.084	0.081	0.088	0.086	0.084	0.083	0.085	0.084	0.08	0.077
Residual tailings SIBX (abs)				0.075				0.072				0.075				0.078				0.08				0.073
Residual concentrate SIBX (mol/l)	6.152E-06	6.15E-06	5.92E-06	5.7E-06	6.46E-06	6.46E-06	6.15E-06	5.62E-06	6.68E-06	6.38E-06	6.38E-06	6E-06	2.22E-05	6.76E-06	6.38E-06	6.15E-06	6.68E-06	6.53E-06	6.38E-06	6.3E-06	6.46E-06	6.38E-06	6.08E-06	5.85E-06
Residual tailings SIBX (mol/l)				5.7E-06				5.47E-06				5.7E-06				5.92E-06				6.08E-06				5.54E-06

Weight of parameter (g)	3SPW Run 1								3SPW Run 2								3SPW Run 3							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	160.51	91.66	39.4	30.71	186.19	71.21	41.2	29.64	170.49	87.31	40.86	33.65	179.62	84.21	48.06	31.83	173.52	73.85	46.08	36.32	160.96	86.49	37.42	29.75
Paper	6.32	6.32	6.29	5.61	6.99	6.97	6.92	5.06	7.23	7.33	7.11	4.5	6.98	6.59	6.57	5	6.76	6.62	6.74	6.64	7.62	7.51	6.79	6.57
B + H_2O	591.46	604.15	596.43	602.59	595.09	599.29	598.46	597.93	597.81	605.96	602.71	605.19	607.95	579.25	597.73	603.68	592.49	597.16	604.29	588.37	605.85	594.8	587.9	600.72
Bottle	515.24	492.02	405.6	443.78	512.63	438.79	375.6	467.56	511.28	480.38	470.84	473.14	494.16	435.86	509.16	498.74	511.12	453.61	467.1	365.01	501.71	452.02	465.68	422.18
D + C + H_2O	1083.41	1032.17	912.25	881.55	1229.57	890.09	876.47	799.53	1140.35	1004.39	869.39	925.15	1213.13	995.41	905.83	882.36	1140.71	909.7	896.36	1005.18	1101.71	1019.31	810.52	867.96
Ore recovered	**154.19**	**85.34**	**33.11**	**25.1**	**179.2**	**64.24**	**34.28**	**24.58**	**163.26**	**79.98**	**33.75**	**29.15**	**172.64**	**77.62**	**41.49**	**26.83**	**166.76**	**67.23**	**39.34**	**29.68**	**153.34**	**78.98**	**30.63**	**23.18**
Water used	**76.22**	**112.13**	**190.83**	**158.81**	**82.46**	**160.5**	**222.86**	**130.37**	**86.53**	**125.58**	**131.87**	**132.05**	**113.79**	**143.39**	**88.57**	**104.94**	**81.37**	**143.55**	**137.19**	**223.36**	**104.14**	**142.78**	**122.22**	**178.54**
Water recovered	**494.53**	**477.09**	**320.64**	**336.86**	**609.44**	**307.74**	**251.66**	**283.8**	**532.09**	**441.22**	**336.1**	**403.17**	**568.23**	**416.79**	**408.1**	**389.81**	**534.11**	**341.31**	**352.16**	**391.36**	**485.76**	**439.94**	**290**	**305.46**
Cumulative ore recovered	154.19	239.53	272.64	297.74	179.2	243.44	277.72	302.3	163.26	243.24	276.99	306.14	172.64	250.26	291.75	318.58	166.76	233.99	273.33	303.01	153.34	232.32	262.95	286.13
Cumulative water recovered	494.53	971.62	1292.26	1629.12	609.44	917.18	1168.84	1452.64	532.09	973.31	1309.41	1712.58	568.23	985.02	1393.12	1782.93	534.11	875.42	1227.58	1618.94	485.76	925.7	1215.7	1521.16
%Cu: concentrates	8.627	0.8906	0.2585	0.1734	6.165	1.351	0.2268	0.2079	6.33	0.7939	0.2273	0.1573	5.67	1.004	0.2254	0.1852	6.238	0.727	0.2056	0.1398	6.189	0.6541	0.2516	0.1814
%Cu: tailings				0.0723				0.0652				0.0578				0.0612				0.0485				0.047
Copper: concentrates (g)	13.301971	0.760038	0.085589	0.043523	11.04768	0.867882	0.077747	0.051102	10.33436	0.634961	0.076714	0.045853	9.788688	0.779305	0.093518	0.049689	10.40249	0.488762	0.080883	0.041493	9.490213	0.516608	0.077065	0.042049
Copper: tailings (g)				0.507734				0.4549				0.401051				0.417029				0.33804				0.335519
Mass cumulative copper (g)	**13.301971**	**14.06201**	**14.1476**	**14.19112**	**11.04768**	**11.91556**	**11.99331**	**12.04441**	**10.33436**	**10.96932**	**11.04603**	**11.09189**	**9.788688**	**10.56799**	**10.66151**	**10.7112**	**10.40249**	**10.89125**	**10.97213**	**11.01363**	**9.490213**	**10.00682**	**10.08389**	**10.12593**
Cumulative copper recovery (%)	90.496643	95.66737	96.24966	96.54576	88.38631	95.32975	95.95176	96.3606	89.91921	95.444	96.11149	96.51046	87.96267	94.96563	95.806	96.25251	91.63843	95.94407	96.65659	97.02211	90.71601	95.65421	96.39087	96.79281
Cumulative copper grade (%)	8.627	5.870667	5.189113	4.76628	6.165	4.894661	4.31849	3.984258	6.33	4.509669	3.987882	3.623142	5.67	4.222805	3.654331	3.36217	6.238	4.65458	4.014244	3.63474	6.189	4.307344	3.834906	3.538928
Cumulative recovered gangue (g)	115.745	198.8884	231.751	256.7252	147.2703	209.002	243.0573	267.4896	133.3919	211.5368	245.065	274.0825	144.349	219.7167	260.9364	287.6228	136.695	202.5124	241.6186	271.1787	125.9116	203.3986	233.8058	256.8643
Residual concentrate SIBX (abs)	0.426	0.374	0.151	0.139	0.14	0.125	0.121	0.121	0.174	0.135	0.129	0.127	0.21	0.175	0.145	0.131	0.183	0.13	0.128	0.125	0.132	0.131	0.128	0.125
Residual tailings SIBX (abs)				0.132				0.119				0.127				0.122				0.122				0.119
Residual concentrate SIBX (mol/l)	2.792E-05	2.45E-05	9.9E-06	9.11E-06	9.18E-06	8.19E-06	7.93E-06	7.93E-06	1.14E-05	8.85E-06	8.46E-06	8.32E-06	1.38E-05	1.15E-05	9.5E-06	8.59E-06	1.2E-05	8.52E-06	8.39E-06	8.19E-06	8.65E-06	8.59E-06	8.39E-06	8.19E-06
Residual tailings SIBX (mol/l)				8.65E-06				7.8E-06				8.32E-06				8E-06				8E-06				7.8E-06

Weight of parameter (g)	5SPW Run 1								5SPW Run 2								5SPW Run 3							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	161.39	83.7	33.23	35.43	174.7	71.4	34.76	27.9	149.49	83.25	32.85	28.58	185.3	70.13	34.13	28.97	168.93	88.88	29.98	25.11	174.09	65.04	33.39	31.11
Paper	5.19	5.13	4.86	4.77	6.76	5.62	4.94	4.85	5.73	5.64	4.65	4.55	6.18	5.6	4.99	4.93	4.93	4.91	4.88	4.75	6.09	6.04	4.83	4.78
B + H2O	605.61	603.39	602.75	609.35	598.13	593.37	602.71	596.33	606.02	609.15	598.14	590.51	598.49	592.63	593.55	598.3	599.7	604.69	596.73	604.46	606.84	602.79	610.99	591.62
Bottle	491.41	530.28	580.16	453.85	464.16	516.44	455.9	417.84	476.94	498.18	356.45	353.79	500.75	497.91	464.03	444.49	525.35	466.94	407.07	293.46	512.78	486.11	450.55	401.45
D + C + H2O	1147.14	975.01	429.84	964.42	1160.43	967.13	837.73	868.35	1074.56	992.78	921.91	941.37	1243.22	912.06	837.31	883.59	1125.02	1099.55	849.51	992.13	1200.17	901.91	846.66	936.56
Ore recovered	**156.2**	**78.57**	**28.37**	**30.66**	**167.94**	**65.78**	**29.82**	**23.05**	**143.76**	**77.61**	**28.2**	**24.03**	**179.12**	**64.53**	**29.14**	**24.04**	**164**	**83.97**	**25.1**	**20.36**	**168**	**59**	**28.56**	**26.33**
Water used	**114.2**	**73.11**	**22.59**	**155.5**	**133.97**	**76.93**	**146.81**	**178.49**	**129.08**	**110.97**	**241.69**	**236.72**	**97.74**	**94.72**	**129.52**	**153.81**	**74.35**	**137.75**	**189.66**	**311**	**94.06**	**116.68**	**160.44**	**190.17**
Water recovered	**518.27**	**465.72**	**11.21**	**417.48**	**500.05**	**466.81**	**293.43**	**306.03**	**443.25**	**446.59**	**284.35**	**319.84**	**607.89**	**395.2**	**310.98**	**344.96**	**528.2**	**520.22**	**267.08**	**299.99**	**579.64**	**368.62**	**289.99**	**359.28**
Cumulative ore recovered	156.2	234.77	263.14	293.8	167.94	233.72	263.54	286.59	143.76	221.37	249.57	273.6	179.12	243.65	272.79	296.83	164	247.97	273.07	293.43	168	227	255.56	281.89
Cumulative water recovered	518.27	983.99	995.2	1412.68	500.05	966.86	1260.29	1566.32	443.25	889.84	1174.19	1494.03	607.89	1003.09	1314.07	1659.03	528.2	1048.42	1315.5	1615.49	579.64	948.26	1238.25	1597.53
%Cu: concentrates	6.669	0.5341	0.2076	0.1411	6.478	0.7503	0.2341	0.1756	6.651	0.7483	0.2293	0.1491	5.758	0.6387	0.2151	0.1496	5.426	0.652	0.2441	0.1811	5.499	0.6102	0.2275	0.1744
%Cu: tailings				0.0628				0.0746				0.0656				0.0642				0.0601				0.0462
Chalcopyrite: concentrates	10.416978	0.419642	0.058896	0.043261	10.87915	0.493547	0.069809	0.040476	9.561478	0.580756	0.064663	0.035829	10.31373	0.412153	0.06268	0.035964	8.89864	0.547484	0.061269	0.036872	9.23832	0.360018	0.064974	0.04592
Chalcopyrite: tailings				0.443494				0.532204				0.476518				0.451435				0.424649				0.331767
Mass cumulative chalcopyrite	**10.416978**	**10.83662**	**10.89552**	**10.93878**	**10.87915**	**11.3727**	**11.44251**	**11.48298**	**9.561478**	**10.14223**	**10.2069**	**10.24272**	**10.31373**	**10.72588**	**10.78856**	**10.82453**	**8.89864**	**9.446124**	**9.507394**	**9.544265**	**9.23832**	**9.598338**	**9.663312**	**9.709232**
Cumulative chalc. recovery (%)	91.519326	95.20613	95.72357	96.10365	90.545	94.6527	95.2337	95.57057	89.19919	94.61707	95.2203	95.55455	91.46652	95.12167	95.67754	95.99648	89.26389	94.7558	95.3704	95.74027	92.00599	95.59147	96.23856	96.69588
Cumulative chalc. grade (%)	6.669	4.615845	4.140578	3.723205	6.478	4.865951	4.341849	4.006764	6.651	4.581575	4.089793	3.743686	5.758	4.402168	3.954897	3.646709	5.426	3.809382	3.481669	3.252655	5.499	4.228343	3.78123	3.444333
Cumulative recovered gangue (g)	126.09313	203.4503	231.6501	262.185	136.4974	200.8509	230.4692	253.4022	116.1257	192.0572	220.0703	243.9967	149.3115	212.6503	241.6092	265.5452	138.2814	220.6691	245.592	265.8454	141.2997	199.2591	227.6314	253.8286
Residual concentrate SIBX (abs)	0.284	0.171	0.17	0.159	0.275	0.18	0.176	0.17	0.23	0.174	0.171	0.167	0.289	0.169	0.165	0.163	0.296	0.171	0.17	0.167	0.295	0.173	0.173	0.169
Residual tailings SIBX (abs)				0.157				0.169				0.163				0.162				0.165				0.169
Residual concentrate SIBX (mol/l)	1.843E-05	1.11E-05	1.1E-05	1.03E-05	1.78E-05	1.17E-05	1.14E-05	1.1E-05	1.49E-05	1.13E-05	1.11E-05	1.08E-05	1.88E-05	1.1E-05	1.07E-05	1.06E-05	1.92E-05	1.11E-05	1.1E-05	1.08E-05	1.91E-05	1.12E-05	1.12E-05	1.1E-05
Residual tailings SIBX (mol/l)				1.02E-05				1.1E-05				1.06E-05				1.05E-05				1.07E-05				1.1E-05

A.7. Synthetic Plant Waters at 100 g/t SIBX, Batch 2

Weight of parameter (g)	1SPW Run 1								1SPW Run 2								1SPW Run 3							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	154.59	49.41	27.5	21.81	151.82	61.67	29.2	18.4	144.53	51.55	34.8	26.83	142.37	56.61	25.77	24.3	161.8	57.51	33.14	23.83	148.43	41.44	33.24	21.73
Paper	6.98	6.2	5.52	4.63	5.11	5.08	4.75	4.66	4.73	4.69	4.56	4.56	4.93	4.85	4.58	4.53	4.78	4.72	4.52	4.38	4.88	4.63	4.6	4.41
B + H_2O	599.64	593.38	608.6	602.29	603.03	607.78	605.54	588.19	605.48	599.53	598.14	605.38	605.64	600.84	603.88	598.74	603.2	603.32	602.82	598.59	600.41	605.78	593.98	600.82
Bottle	497.74	407.62	390.31	344.22	485.41	473.81	428.9	319.03	493.34	415.11	412.44	369.75	517.14	525.87	395.07	447.01	496.27	462.15	378.37	379.13	495.17	445.61	359.94	319.77
D + C + H_2O	1051.82	802.26	756.75	791.12	1052.68	841.35	756.51	775.25	1015.07	839.55	833.5	870.08	996.03	754.07	750.33	724.96	1116.67	874.43	874.53	829.76	1048.49	750.24	854.71	828.78
Ore recovered	**147.61**	**43.21**	**21.98**	**17.18**	**146.71**	**56.59**	**24.45**	**13.74**	**139.8**	**46.86**	**30.24**	**22.27**	**137.44**	**51.76**	**21.19**	**19.77**	**157.02**	**52.79**	**28.62**	**19.45**	**143.55**	**36.81**	**28.64**	**17.32**
Water used	**101.9**	**185.76**	**218.29**	**258.07**	**117.62**	**133.97**	**176.64**	**269.16**	**112.14**	**184.42**	**185.7**	**235.63**	**88.5**	**74.97**	**208.81**	**151.73**	**106.93**	**141.17**	**224.45**	**219.46**	**105.24**	**160.17**	**234.04**	**281.05**
Water recovered	**443.84**	**215.68**	**148.81**	**155.09**	**429.88**	**293.18**	**187.75**	**131.57**	**404.66**	**250.66**	**249.89**	**251.4**	**411.62**	**269.73**	**152.66**	**192.68**	**494.25**	**322.86**	**253.79**	**230.07**	**441.23**	**195.65**	**224.36**	**169.63**
Cumulative ore recovered	147.61	190.82	212.8	229.98	146.71	203.3	227.75	241.49	139.8	186.66	216.9	239.17	137.44	189.2	210.39	230.16	157.02	209.81	238.43	257.88	143.55	180.36	209	226.32
Cumulative water recovered	443.84	659.52	808.33	963.42	429.88	723.06	910.81	1042.38	404.66	655.32	905.21	1156.61	411.62	681.35	834.01	1026.69	494.25	817.11	1070.9	1300.97	441.23	636.88	861.24	1030.87
%Cu: concentrates	8.158	0.8538	0.308	0.2101	7.54	0.5949	0.2499	0.2089	7.547	0.6901	0.2094	0.1539	9.011	0.4966	0.2395	0.1651	7.539	0.5156	0.2255	0.1542	7.064	0.509	0.2114	0.1651
%Cu: tailings				0.0581				0.0641				0.0548				0.0674				0.0449				0.0403
Copper: concentrates (g)	12.04202	0.368927	0.067698	0.036095	11.06193	0.336654	0.061101	0.028703	10.55071	0.323381	0.063323	0.034274	12.38472	0.25704	0.05075	0.03264	11.83774	0.272185	0.064538	0.029992	10.14037	0.187363	0.060545	0.028595
Copper: tailings (g)				0.447382				0.486205				0.416935				0.518872				0.333212				0.311793
Mass cumulative copper (g)	**12.04202**	**12.41095**	**12.47865**	**12.51474**	**11.06193**	**11.39859**	**11.45969**	**11.48839**	**10.55071**	**10.87409**	**10.93741**	**10.97168**	**12.38472**	**12.64176**	**12.69251**	**12.72515**	**11.83774**	**12.10992**	**12.17446**	**12.20445**	**10.14037**	**10.32773**	**10.38828**	**10.41688**
Cumulative copper recovery (%)	92.90161	95.7478	96.27008	96.54855	92.37835	95.18975	95.7	95.9397	92.64255	95.48206	96.03808	96.33902	93.51177	95.45257	95.83576	96.08222	94.4174	96.58834	97.1031	97.34231	94.5166	96.26297	96.8273	97.09383
Cumulative copper grade (%)	8.158	6.504009	5.864027	5.441666	7.54	5.606782	5.031696	4.757295	7.547	5.825612	5.042605	4.587399	9.011	6.681691	6.032848	5.528827	7.539	5.771852	5.106095	4.732609	7.064	5.726178	4.970469	4.60272
Cumulative recovered gangue (g)	112.8065	154.9502	176.7345	193.8102	114.7391	170.3561	194.6295	208.2866	109.3066	155.232	185.289	207.4599	101.646	152.6631	173.7064	193.3821	122.8069	174.8102	203.2437	222.607	114.2426	150.5111	178.9761	196.2134
Residual concentrate SIBX (abs)	0.279	0.084	0.083	0.082	0.292	0.089	0.084	0.081	0.269	0.093	0.085	0.079	0.236	0.08	0.068	0.082	0.279	0.092	0.083	0.082	0.169	0.155	0.105	0.092
Residual tailings SIBX (abs)				0.077				0.078				0.069				0.077				0.071				0.083
Residual concentrate SIBX (ppm)	1.94E-05	5.85E-06	5.78E-06	5.71E-06	2.03E-05	6.2E-06	5.85E-06	5.64E-06	1.87E-05	6.47E-06	5.92E-06	5.5E-06	1.64E-05	5.57E-06	4.73E-06	5.71E-06	1.94E-05	6.4E-06	5.78E-06	5.71E-06	1.18E-05	1.08E-05	7.31E-06	6.4E-06
Residual tailings SIBX (ppm)				5.36E-06				5.43E-06				4.8E-06				5.36E-06				4.94E-06				5.78E-06

Weight of parameter (g)	3SPW Run 1								3SPW Run 2								3SPW Run 3							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	170.4	60.7	46.11	15.52	153.48	56.94	33.14	27.34	176.63	57.7	33.64	25.51	166.86	54.77	33.72	29.25	168.06	51.47	39.83	27.2	198.29	58.65	33.25	34.16
Paper	7.44	8.46	5.03	4.63	5.59	5.53	4.74	4.7	9.69	5.77	5.61	4.5	8.87	6.03	5.54	4.74	9.9	5.83	5.81	4.79	9.63	8.95	6.11	6
B + H_2O	576.65	596.18	601.56	575.9	508.07	443.06	478.46	458.2	590.02	605.15	606.54	609.04	596.44	600.64	607.72	600.22	598.01	584.54	605.52	603.99	521.99	490.98	476.78	457.26
Bottle	460.5	461.57	489.98	489.09	407.17	334.15	356.82	327.9	491.08	498.87	517.68	504.5	521.99	490.98	476.78	457.26	508.07	443.06	478.46	458.2	430.59	408.91	418.59	264.82
D + C + H_2O	1158.81	795.74	773.52	677.13	1036.88	824.46	761.55	777.16	1143.54	843.13	747.03	748.57	1070.29	816.42	794.25	838.16	1079.21	794.34	827.66	791.77	1232.3	811.41	721.96	752.16
Ore recovered	**162.96**	**52.24**	**41.08**	**10.89**	**147.89**	**51.41**	**28.4**	**22.64**	**166.94**	**51.93**	**28.03**	**21.01**	**157.99**	**48.74**	**28.18**	**24.51**	**158.16**	**45.64**	**34.02**	**22.41**	**188.66**	**49.7**	**27.14**	**28.16**
Water used	**116.15**	**134.61**	**111.58**	**86.81**	**100.9**	**108.91**	**121.64**	**130.3**	**98.94**	**106.28**	**88.86**	**104.54**	**74.45**	**109.66**	**130.94**	**142.96**	**89.94**	**141.48**	**127.06**	**145.79**	**91.4**	**82.07**	**58.19**	**192.44**
Water recovered	**521.23**	**251.28**	**253.19**	**218.65**	**429.62**	**306.53**	**243.84**	**263.44**	**519.19**	**327.31**	**262.47**	**262.24**	**479.38**	**300.41**	**267.46**	**309.91**	**472.64**	**249.61**	**298.91**	**262.79**	**593.77**	**322.03**	**268.96**	**170.78**
Cumulative ore recovered	162.96	215.2	256.28	267.17	147.89	199.3	227.7	250.34	166.94	218.87	246.9	267.91	157.99	206.73	234.91	259.42	158.16	203.8	237.82	260.23	188.66	238.36	265.5	293.66
Cumulative water recovered	521.23	772.51	1025.7	1244.35	429.62	736.15	979.99	1243.43	519.19	846.5	1108.97	1371.21	479.38	779.79	1047.25	1357.16	472.64	722.25	1021.16	1283.95	593.77	915.8	1184.76	1355.54
%Cu: concentrates	5.47	1.023	0.3258	0.189	7.982	0.5548	0.281	0.1772	6.199	0.4655	0.2114	0.1266	7.209	0.7689	0.2767	0.1792	7.942	0.6542	0.217	0.1565	5.04	0.5263	0.2274	0.1436
%Cu: tailings				0.0556				0.0515				0.0439				0.0506				0.0437				0.0392
Copper: concentrates (g)	8.913912	0.534415	0.133839	0.020582	11.80458	0.285223	0.079804	0.040118	10.34861	0.241734	0.059255	0.026599	11.3895	0.374762	0.077974	0.043922	12.56107	0.298577	0.073823	0.035072	9.508464	0.261571	0.061716	0.040438
Copper: tailings (g)				0.407453				0.386075				0.321388				0.374733				0.323279				0.276885
Mass cumulative copper (g)	**8.913912**	**9.448327**	**9.582166**	**9.602748**	**11.80458**	**12.0898**	**12.16961**	**12.20972**	**10.34861**	**10.59034**	**10.6496**	**10.6762**	**11.3895**	**11.76426**	**11.84224**	**11.88616**	**12.56107**	**12.85964**	**12.93347**	**12.96854**	**9.508464**	**9.770035**	**9.831751**	**9.872189**
Cumulative copper recovery (%)	89.04828	94.38698	95.72401	95.92962	93.71838	95.98281	96.61639	96.93489	94.09893	96.29699	96.8358	97.07765	92.89292	95.94948	96.58544	96.94367	94.50225	96.74857	97.30397	97.56783	93.68799	96.26528	96.87338	97.27182
Cumulative copper grade (%)	5.47	4.390487	3.738944	3.594246	7.982	6.066133	5.344579	4.877257	6.199	4.838646	4.313325	3.984995	7.209	5.69064	5.04118	4.58182	7.942	6.309933	5.438343	4.983491	5.04	4.098857	3.703108	3.361775
Cumulative recovered gangue (g)	137.1972	187.8927	228.5859	239.4164	113.7727	164.3584	192.5277	215.0518	137.0307	188.2621	216.1208	237.0539	125.0724	172.7292	200.6839	225.0669	121.8563	166.6334	200.44	222.7487	161.1789	210.1229	237.0845	265.1277
Residual concentrate SIBX (abs)	0.409	0.127	0.124	0.124	0.42	0.132	0.127	0.12	0.52	0.129	0.128	0.128	0.511	0.128	0.122	0.121	0.452	0.13	0.128	0.123	0.475	0.126	0.123	0.122
Residual tailings SIBX (abs)				0.121				0.117				0.126				0.119				0.121				0.119
Residual concentrate SIBX (mol/l)	2.68E-05	8.32E-06	8.13E-06	8.13E-06	2.75E-05	8.65E-06	8.32E-06	7.87E-06	3.41E-05	8.46E-06	8.39E-06	8.39E-06	3.35E-05	8.39E-06	8E-06	7.93E-06	2.96E-05	8.52E-06	8.39E-06	8.06E-06	3.11E-05	8.26E-06	8.06E-06	8E-06
Residual tailings SIBX (mol/l)				7.93E-06				7.67E-06				8.26E-06				7.8E-06				7.93E-06				7.8E-06

Weight of parameter (g)	5SPW Run 1								5SPW Run 2								5SPW Run 3							
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
C + Paper	140.5	56.26	31.99	30.69	146.77	56.05	35.74	26.57	159.17	49.3	32.69	27.24	146.41	85.31	26.04	27.07	128.96	63.57	38.32	30.05	150.31	59.95	38.88	23.77
Paper	4.83	4.64	4.59	4.59	6.09	4.84	4.82	4.7	4.77	4.73	4.65	4.42	6.05	5.15	4.44	4.43	4.72	4.72	4.6	4.58	6.33	5.41	5.4	4.61
B + H2O	597.07	600.73	604.33	600.72	599.13	605.49	596.55	592.32	603.46	604	600.58	607.81	592.87	609.4	598.95	596.9	598.01	597.21	607.42	604.19	602.69	590.46	591.35	608.91
Bottle	493.61	507.68	371.87	431.35	480.41	489.23	411.94	390.94	502.96	443.31	424.62	348.22	496.5	496.85	350.15	487.62	469.49	463.12	359.23	320.76	422.51	536.53	436.59	346.91
D + C + H2O	993.79	826.85	886.3	877.2	1028.7	852.18	899.85	869.25	1105.27	831.09	853.54	934.84	1034.69	1096.86	813.79	864.73	983.74	936.61	1008.56	1017.5	1011.21	878.12	895.23	901.99
Ore recovered	**135.67**	**51.62**	**27.4**	**26.1**	**140.68**	**51.21**	**30.92**	**21.87**	**154.4**	**44.57**	**28.04**	**22.82**	**140.36**	**80.16**	**21.6**	**22.64**	**124.24**	**58.85**	**33.72**	**25.47**	**143.98**	**54.54**	**33.48**	**19.16**
Water used	**103.46**	**93.05**	**232.46**	**169.37**	**118.72**	**116.26**	**184.61**	**201.38**	**100.5**	**160.69**	**175.96**	**259.59**	**96.37**	**112.55**	**248.8**	**109.28**	**128.52**	**134.09**	**248.19**	**283.43**	**180.18**	**53.93**	**154.76**	**262**
Water recovered	**396.19**	**324.57**	**258.77**	**320.95**	**410.83**	**327.1**	**316.65**	**285.22**	**491.9**	**268.22**	**281.87**	**291.65**	**439.49**	**546.54**	**175.72**	**372.03**	**372.51**	**386.06**	**358.98**	**347.82**	**328.58**	**412.04**	**339.32**	**260.05**
Cumulative ore recovered	135.67	187.29	214.69	240.79	140.68	191.89	222.81	244.68	154.4	198.97	227.01	249.83	140.36	220.52	242.12	264.76	124.24	183.09	216.81	242.28	143.98	198.52	232	251.16
Cumulative water recovered	396.19	720.76	979.53	1300.48	410.83	737.93	1054.58	1339.8	491.9	760.12	1041.99	1333.64	439.49	986.03	1161.75	1533.78	372.51	758.57	1117.55	1465.37	328.58	740.62	1079.94	1339.99
%Cu: concentrates	8.538	0.5412	0.2492	0.1852	10.24	0.4908	0.1937	0.1407	6.269	0.4155	0.1782	0.137	8.023	0.4028	0.1988	0.1443	10.05	0.4509	0.1821	0.1277	7.877	0.4419	0.2146	0.1563
%Cu: tailings				0.0554				0.0567				0.0493				0.0569				0.0422				0.0451
Coppere: concentrates (g)	11.5835	0.279367	0.068281	0.048337	14.40563	0.251339	0.059892	0.030771	9.679336	0.185188	0.049967	0.031263	11.26108	0.322884	0.042941	0.03267	12.48612	0.265355	0.061404	0.032525	11.3413	0.241012	0.071848	0.029947
Copper: tailings (g)				0.420602				0.428266				0.369834				0.418352				0.319758				0.337727
Mass cumulative copper (g)	**11.5835**	**11.86287**	**11.93115**	**11.97949**	**14.40563**	**14.65697**	**14.71686**	**14.74763**	**9.679336**	**9.864524**	**9.914492**	**9.945755**	**11.26108**	**11.58397**	**11.62691**	**11.65958**	**12.48612**	**12.75147**	**12.81288**	**12.8454**	**11.3413**	**11.58232**	**11.65416**	**11.68411**
Cumulative copper recovery (%)	93.41466	95.66761	96.21826	96.60807	94.9244	96.58057	96.97522	97.17798	93.83212	95.62735	96.11174	96.41481	93.23687	95.91021	96.26574	96.53623	94.84213	96.85771	97.32413	97.57118	94.33918	96.34397	96.94162	97.19072
Cumulative copper grade (%)	8.538	6.333959	5.557386	4.975078	10.24	7.638215	6.605118	6.027315	6.269	4.957795	4.367425	3.981009	8.023	5.253023	4.802126	4.403829	10.05	6.964594	5.909727	5.301884	7.877	5.834332	5.023347	4.652059
Cumulative recovered gangue (g)	102.1917	153.0042	180.2069	206.1672	99.04523	149.5288	180.2757	202.0568	126.425	170.4598	198.3554	221.085	107.8135	187.0403	208.5162	231.0618	88.15295	146.236	179.7786	205.1546	111.2017	165.0451	198.3174	217.3909
Residual concentrate SIBX (abs)	0.52	0.164	0.165	0.164	0.505	0.1905	0.169	0.159	0.53352	0.168264	0.16929	0.168264	0.536	0.1865	0.1685	0.166	0.550593	0.173648	0.174707	0.173648	0.582	0.1955	0.1745	0.1655
Residual tailings SIBX (abs)				0.155				0.158				0.159				0.165				0.164				0.161
Residual concentrate SIBX (ppm)	3.37E-05	1.06E-05	1.07E-05	1.06E-05	3.28E-05	1.24E-05	1.1E-05	1.03E-05	3.46E-05	1.09E-05	1.1E-05	1.09E-05	3.48E-05	1.21E-05	1.09E-05	1.08E-05	3.57E-05	1.13E-05	1.13E-05	1.13E-05	3.78E-05	1.27E-05	1.13E-05	1.07E-05
Residual tailings SIBX (ppm)				1.01E-05				1.03E-05				1.03E-05				1.07E-05				1.06E-05				1.04E-05

Appendix B: Kevitsa Ore Flotation and Adsorption Tests

B.1. All Synthetic and Actual Plant Waters at 0 g/t SIPX, 0 g/t Aerophine

Weight of parameter (g)	17: Process water_1							18: Process water_2							15: Cu thickener overflow_1						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	22.59	27.26	35.6	18.24	20.74	20.43	29.07	24.42	20.35	31	20.17	23.67	30.35	30.37	23.17	18.13	25.54	18.48	21.66	22.44	26.14
Paper	7.66	7.8	7.22	7.78	7.23	7.86	7.72	7.69	7.77	7.35	7.23	7.74	7.37	7.29	7.36	7.39	7.46	7.79	7.73	7.38	7.82
D + C + H_2O	666.3	609.7	939.2	694.6	798.3	882	1124.1	672.7	608.4	840.6	726.2	844.4	969	1182.5	658.9	566.8	702.1	665.7	755.2	714	891.2
Ore recovered	**14.93**	**19.46**	**28.38**	**10.46**	**13.51**	**12.57**	**21.35**	**16.73**	**12.58**	**23.65**	**12.94**	**15.93**	**22.98**	**23.08**	**15.81**	**10.74**	**18.08**	**10.69**	**13.93**	**15.06**	**18.32**
Water recovered	**110.67**	**138.74**	**466.42**	**145.04**	**244.39**	**428.53**	**563.45**	**115.27**	**144.32**	**372.55**	**174.16**	**288.07**	**505.12**	**620.12**	**102.39**	**104.56**	**239.62**	**115.91**	**200.87**	**258.04**	**333.58**
Cumulative ore recovered	14.93	34.39	62.77	73.23	86.74	99.31	120.66	16.73	29.31	52.96	65.9	81.83	104.81	127.89	15.81	26.55	44.63	55.32	69.25	84.31	102.63
Cumulative water recovered	110.67	249.41	715.83	860.87	1105.26	1533.79	2097.24	115.27	259.59	632.14	806.3	1094.37	1599.49	2219.61	102.39	206.95	446.57	562.48	763.35	1021.39	1354.97
%Cu: concentrates	9.56	3.65	1.27	1.74	0.59	0.36	0.26	9.29	3.32	1.34	1.55	0.52	0.3	0.2	9.49	5.17	2.3	2.91	1.17	0.99	1
%Cu: tailings							0.07							0.07							0.12
%Ni: concentrates	0.67	0.75	0.77	0.89	0.76	0.56	0.42	0.8	0.83	0.79	0.92	0.78	0.54	0.39	0.55	0.63	0.58	0.7	0.59	0.51	0.4
%Ni: tailings							0.16							0.15							0.18
Chalcopyrite: concentrates	4.125168	2.052861	1.041694	0.526023	0.230373	0.130786	0.160434	4.491957	1.207098	0.915925	0.579682	0.23941	0.199249	0.13341	4.336327	1.604792	1.20185	0.899072	0.471043	0.430908	0.52948
Chalcopyrite: tailings							1.779012							1.764384							3.112266
pentlandite: concentrates	0.243384	0.355109	0.531693	0.226506	0.24982	0.17127	0.218175	0.325645	0.254049	0.454586	0.289655	0.302321	0.301927	0.219007	0.211569	0.164628	0.255144	0.182068	0.199968	0.186876	0.178297
pentlandite: tailings							3.423221							3.182883							3.930088
Mass cumulative chalcopyrite	**4.125168**	**6.178029**	**7.219723**	**7.745746**	**7.976118**	**8.106905**	**8.267338**	**4.491957**	**5.699055**	**6.61498**	**7.194662**	**7.434072**	**7.633321**	**7.766731**	**4.336327**	**5.941118**	**7.142968**	**8.04204**	**8.513084**	**8.943991**	**9.473471**
Mass cumulative pentlandite	**0.243384**	**0.598494**	**1.130187**	**1.356693**	**1.606513**	**1.777783**	**1.995959**	**0.325645**	**0.579693**	**1.03428**	**1.323934**	**1.626255**	**1.928182**	**2.14719**	**0.211569**	**0.376197**	**0.631341**	**0.813409**	**1.013377**	**1.200253**	**1.37855**
Cumulative chalc. recovery (%)	41.06136	61.49526	71.86414	77.1001	79.3932	80.69503	82.29196	47.12939	59.79421	69.40405	75.48604	77.99792	80.08843	81.48817	34.45429	47.20517	56.75447	63.89805	67.64073	71.0645	75.27148
Cumulative pent. recovery (%)	4.491167	11.04399	20.85532	25.03503	29.64495	32.8054	36.83138	6.109574	10.8759	19.40461	24.83895	30.51094	36.17554	40.28443	3.985379	7.086509	11.89271	15.32236	19.08921	22.60944	25.96805
Cumulative chalc. grade (%)	27.63006	17.96461	11.50187	10.57728	9.195433	8.163231	6.851764	26.84971	19.44406	12.49052	10.91754	9.084776	7.283008	6.072978	27.42775	22.37709	16.00486	14.53731	12.29326	10.60846	9.230704
Cumulative pent. grade (%)	1.63017	1.740314	1.800522	1.852647	1.852102	1.790135	1.654201	1.946472	1.977801	1.952945	2.009005	1.987359	1.839693	1.678935	1.3382	1.416938	1.41461	1.47037	1.46336	1.423619	1.343223

Weight of parameter (g)	16: Cu thickener overflow_2							19: Ni thickener overflow_1							20: Ni thickener overflow_2						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	23.38	19.22	32.66	18.96	18.9	25.88	30.04	17.24	23.26	24.26	15.72	21.36	24.05	27.31	17.09	15.4	23.26	17.14	20.18	23.01	25.25
Paper	7.58	7.71	7.65	7.42	7.72	7.53	7.6	7.72	7.28	7.73	7.52	7.42	7.64	7.37	7.22	7.74	7.31	7.33	7.73	7.88	7.79
D + C + H_2O	650.4	560.7	767.4	679.2	709	734.7	943	602.3	576.7	611.2	614.8	692.4	650.8	828.4	601.7	511.1	595.5	623.7	671.8	619.7	781.6
Ore recovered	**15.8**	**11.51**	**25.01**	**11.54**	**11.18**	**18.35**	**22.44**	**9.52**	**15.98**	**16.53**	**8.2**	**13.94**	**16.41**	**19.94**	**9.87**	**7.66**	**15.95**	**9.81**	**12.45**	**15.13**	**17.46**
Water recovered	**93.9**	**97.69**	**297.99**	**128.56**	**157.42**	**275.45**	**381.26**	**52.08**	**109.22**	**150.27**	**67.5**	**138.06**	**193.49**	**269.16**	**51.13**	**51.94**	**135.15**	**74.79**	**118.95**	**163.67**	**224.84**
Cumulative ore recovered	15.8	27.31	52.32	63.86	75.04	93.39	115.83	9.52	25.5	42.03	50.23	64.17	80.58	100.52	9.87	17.53	33.48	43.29	55.74	70.87	88.33
Cumulative water recovered	93.9	191.59	489.58	618.14	775.56	1051.01	1432.27	52.08	161.3	311.57	379.07	517.13	710.62	979.78	51.13	103.07	238.22	313.01	431.96	595.63	820.47
%Cu: concentrates	9.45	6.21	2.81	2.55	2.27	1.93	1.01	5.99	3.64	2.31	3.11	1.15	0.7	0.45	6.31	5.52	2.84	3.51	1.47	0.76	0.52
%Cu: tailings							0.17							0.1							0.1
%Ni: concentrates	0.45	0.45	0.41	0.52	0.46	0.5	0.55	1.01	1.02	0.95	1.34	0.83	0.62	0.45	1.06	1.28	1.07	1.62	1.07	0.69	0.49
%Ni: tailings							0.18							0.15							0.16
Chalcopyrite: concentrates	4.315318	2.065812	2.031159	0.850491	0.733486	1.023569	0.65504	1.648116	1.681133	1.103592	0.737052	0.463324	0.331994	0.259335	1.799991	1.222058	1.309191	0.995176	0.528945	0.332335	0.262405
Chalcopyrite: tailings							4.344188							2.599653							2.634884
pentlandite: concentrates	0.172993	0.126022	0.249491	0.146005	0.125129	0.223236	0.300292	0.233946	0.396584	0.38208	0.267348	0.281513	0.247547	0.218321	0.254555	0.23856	0.415243	0.386672	0.324124	0.254007	0.208161
pentlandite: tailings							3.872277							3.282774							3.54908
Mass cumulative chalcopyrite	**4.315318**	**6.38113**	**8.412289**	**9.26278**	**9.996266**	**11.01984**	**11.67488**	**1.648116**	**3.329249**	**4.432841**	**5.169893**	**5.633217**	**5.965211**	**6.224546**	**1.799991**	**3.022049**	**4.33124**	**5.326416**	**5.855361**	**6.187697**	**6.450101**
Mass cumulative pentlandite	**0.172993**	**0.299015**	**0.548506**	**0.694511**	**0.81964**	**1.042876**	**1.343168**	**0.233946**	**0.63053**	**1.012611**	**1.279959**	**1.561472**	**1.809019**	**2.027341**	**0.254555**	**0.493114**	**0.908358**	**1.295029**	**1.619153**	**1.873161**	**2.081321**
Cumulative chalc. recovery (%)	26.93864	39.8346	52.51424	57.82348	62.40231	68.79201	72.88114	18.67723	37.72862	50.23505	58.58767	63.83828	67.60059	70.5395	19.81281	33.26422	47.6747	58.62878	64.45097	68.10904	70.99737
Cumulative pent. recovery (%)	3.316931	5.733252	10.51696	13.31643	15.71563	19.99591	25.75366	4.405677	11.87414	19.06947	24.10416	29.40562	34.06743	38.17885	4.521076	8.758067	16.13309	23.00065	28.75733	33.26869	36.96577
Cumulative chalc. grade (%)	27.31214	23.36554	16.07853	14.50482	13.32125	11.7998	10.07932	17.31214	13.05588	10.54685	10.29244	8.778583	7.402843	6.192346	18.23699	17.2393	12.9368	12.30403	10.50477	8.731052	7.302277
Cumulative pent. grade (%)	1.094891	1.094891	1.048368	1.087552	1.092271	1.116689	1.159603	2.457421	2.472668	2.409257	2.548196	2.433336	2.244998	2.016853	2.579075	2.812974	2.713135	2.99152	2.904832	2.643094	2.356302

Weight of parameter (g)	21: Mine Pit waters_1							22: Mine pit waters_2							49: Raw water_1						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	24.7	19.24	31.65	20.38	26.31	33.01	34.29	19.02	21.68	33.75	23.06	22.48	30.13	33.48	22.36	35.12	79.49	28.54	36.59	34.59	35.28
Paper	7.76	7.53	7.67	7.5	7.81	7.59	7.42	7.27	7.82	7.28	7.55	7.56	7.73	7.71	7.59	7.3	7.43	7.34	7.32	7.33	7.28
D + C + H_2O	692.6	602.3	865.5	727.4	900.7	1040	1321.3	642.3	626	913.7	774.5	837.4	965.6	1263.3	685.4	831.9	1678.3	907.4	1260.1	1357.3	1516.6
Ore recovered	**16.94**	**11.71**	**23.98**	**12.88**	**18.5**	**25.42**	**26.87**	**11.75**	**13.86**	**26.47**	**15.51**	**14.92**	**22.4**	**25.77**	**14.77**	**27.82**	**72.06**	**21.2**	**29.27**	**27.26**	**28**
Water recovered	**134.96**	**139.09**	**397.12**	**175.42**	**341.8**	**573.68**	**755.13**	**89.85**	**160.64**	**442.83**	**219.89**	**282.08**	**502.3**	**698.23**	**129.93**	**352.58**	**1161.84**	**347.1**	**690.43**	**889.14**	**949.3**
Cumulative ore recovered	16.94	28.65	52.63	65.51	84.01	109.43	136.3	11.75	25.61	52.08	67.59	82.51	104.91	130.68	14.77	42.59	114.65	135.85	165.12	192.38	220.38
Cumulative water recovered	134.96	274.05	671.17	846.59	1188.39	1762.07	2517.2	89.85	250.49	693.32	913.21	1195.29	1697.59	2395.82	129.93	482.51	1644.35	1991.45	2681.88	3571.02	4520.32
%Cu: concentrates	6.78	4.07	1.54	1.88	0.62	0.33	0.22	8.95	4.26	1.38	1.9	0.62	0.36	0.24	6.5831	2.8791	0.7345	1.377	0.5788	0.3484	0.2025
%Cu: tailings							0.08							0.08							0.1647
%Ni: concentrates	0.44	0.43	0.38	0.46	0.41	0.36	0.32	0.45	0.44	0.38	0.46	0.4	0.35	0.32	0.4544	0.4276	0.3561	0.4078	0.3663	0.3307	0.2769
%Ni: tailings							0.22							0.24							0.2355
Chalcopyrite: concentrates	3.319457	1.377448	1.067318	0.699838	0.331503	0.242445	0.17085	3.039379	1.706462	1.05574	0.851705	0.267353	0.233064	0.178751	2.810185	2.31493	1.529713	0.843711	0.489638	0.274491	0.163873
Chalcopyrite: tailings							1.996994							2.009988							3.711081
pentlandite: concentrates	0.181353	0.122513	0.221713	0.144156	0.18455	0.222657	0.209207	0.12865	0.14838	0.244735	0.173591	0.145207	0.190754	0.200642	0.163297	0.289436	0.624345	0.210349	0.260866	0.21934	0.188642
pentlandite: tailings							4.623212							5.076321							4.467166
Mass cumulative chalcopyrite	**3.319457**	**4.696905**	**5.764223**	**6.464061**	**6.795564**	**7.038009**	**7.208858**	**3.039379**	**4.745841**	**5.801581**	**6.653286**	**6.920639**	**7.153702**	**7.332454**	**2.810185**	**5.125114**	**6.654827**	**7.498538**	**7.988176**	**8.262667**	**8.42654**
Mass cumulative pentlandite	**0.181353**	**0.303866**	**0.525579**	**0.669735**	**0.854285**	**1.076942**	**1.286148**	**0.12865**	**0.277029**	**0.521764**	**0.695355**	**0.840562**	**1.031316**	**1.231959**	**0.163297**	**0.452733**	**1.077078**	**1.287427**	**1.548293**	**1.767633**	**1.956276**
Cumulative chalc. recovery (%)	36.05811	51.02085	62.61476	70.21686	73.81786	76.45146	78.30734	32.53302	50.79872	62.09919	71.21571	74.0774	76.57208	78.48541	23.15268	42.22503	54.8281	61.77931	65.81336	68.07485	69.42497
Cumulative pent. recovery (%)	3.068908	5.142116	8.89401	11.33346	14.45647	18.22434	21.7646	2.039377	4.391517	8.271098	11.0229	13.32474	16.34861	19.52923	2.542197	7.048135	16.76792	20.04264	24.10379	27.51848	30.45526
Cumulative chalc. grade (%)	19.59538	16.39408	10.95235	9.867288	8.088994	6.431517	5.288964	25.86705	18.5312	11.13975	9.843595	8.387636	6.818895	5.610999	19.0263	12.03361	5.804472	5.519719	4.8378	4.294972	3.823641
Cumulative pent. grade (%)	1.07056	1.060615	0.99863	1.02234	1.016885	0.984137	0.943616	1.094891	1.081723	1.001851	1.028784	1.01874	0.983049	0.942729	1.105596	1.063003	0.939448	0.947683	0.937678	0.918824	0.887683

Weight of parameter (g)	50: Raw water_2						
	C1	C2	C3	C4	C5	C6	C7
C + Paper	27.41	36.61	60.91	30.92	31.35	32.13	40.25
Paper	7.39	7.41	7.42	7.49	7.48	7.4	7.37
D + C + H_2O	758.9	870.2	1503.9	966.7	1166.1	1292.1	1758.4
Ore recovered	**20.02**	**29.2**	**53.49**	**23.43**	**23.87**	**24.73**	**32.88**
Water recovered	**198.18**	**389.5**	**1006.01**	**404.17**	**601.83**	**826.47**	**1186.22**
Cumulative ore recovered	20.02	49.22	102.71	126.14	150.01	174.74	207.62
Cumulative water recovered	198.18	587.68	1593.69	1997.86	2599.69	3426.16	4612.38
%Cu: concentrates	4.9335	2.3003	0.8415	1.2264	0.5327	0.3497	0.2
%Cu: tailings							0.1369
%Ni: concentrates	0.425	0.3957	0.3498	0.3802	0.3498	0.3195	0.2687
%Ni: tailings							0.1961
Chalcopyrite: concentrates	2.854586	1.941294	1.30092	0.830478	0.367501	0.249945	0.190058
Chalcopyrite: tailings							3.135168
pentlandite: concentrates	0.207019	0.28113	0.455251	0.216742	0.203156	0.192244	0.21496
pentlandite: tailings							3.780674
Mass cumulative chalcopyrite	**2.854586**	**4.795879**	**6.0968**	**6.927278**	**7.294779**	**7.544724**	**7.734782**
Mass cumulative pentlandite	**0.207019**	**0.488149**	**0.9434**	**1.160142**	**1.363298**	**1.555542**	**1.770502**
Cumulative chalc. recovery (%)	26.26126	44.12053	56.08857	63.7287	67.10959	69.409	71.15747
Cumulative pent. recovery (%)	3.72929	8.79362	16.9946	20.89902	24.55872	28.02185	31.89418
Cumulative chalc. grade (%)	14.25867	9.743762	5.935936	5.491738	4.862862	4.317686	3.725451
Cumulative pent. grade (%)	1.034063	0.99177	0.918508	0.919726	0.908805	0.890204	0.852761

Weight of parameter (g)	27: 1SPW_1							28: 1SPW_2							29: 3SPW_1						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	26.18	18.16	37.53	19.72	27.12	30.57	34.16	23.59	23.97	43.68	23.67	28.34	33.07	28.26	22.05	25.11	43.44	22.6	27.99	33.12	36.19
Paper	7.57	7.62	7.63	7.58	7.44	7.73	7.56	7.55	7.57	7.42	7.7	7.59	7.61	7.4	7.76	7.48	7.6	7.63	7.43	7.56	7.45
D + C + H2O	709.6	590.4	981.1	729.8	942.4	1016.4	1344.6	694.4	665.9	1083.7	789.9	970.2	1091	1229.9	667.1	676.6	1076.6	767.8	963.9	1090.6	1429.6
Ore recovered	**18.61**	**10.54**	**29.9**	**12.14**	**19.68**	**22.84**	**26.6**	**16.04**	**16.4**	**36.26**	**15.97**	**20.75**	**25.46**	**20.86**	**14.29**	**17.63**	**35.84**	**14.97**	**20.56**	**25.56**	**28.74**
Water recovered	**150.29**	**128.36**	**506.8**	**178.56**	**382.32**	**552.66**	**778.7**	**137.66**	**198**	**603.04**	**234.83**	**409.05**	**624.64**	**669.74**	**112.11**	**207.47**	**596.36**	**213.73**	**402.94**	**624.14**	**861.56**
Cumulative ore recovered	18.61	29.15	59.05	71.19	90.87	113.71	140.31	16.04	32.44	68.7	84.67	105.42	130.88	151.74	14.29	31.92	67.76	82.73	103.29	128.85	157.59
Cumulative water recovered	150.29	278.65	785.45	964.01	1346.33	1898.99	2677.69	137.66	335.66	938.7	1173.53	1582.58	2207.22	2876.96	112.11	319.58	915.94	1129.67	1532.61	2156.75	3018.31
%Cu: concentrates	6.58	4.71	1.61	2.24	0.61	0.37	0.24	7.7	3.01	1.13	1.72	0.55	0.36	0.27	8.41	3.47	1.09	2.03	0.52	0.28	0.19
%Cu: tailings							0.1							0.12							0.09
%Ni: concentrates	0.43	0.44	0.38	0.44	0.38	0.34	0.31	0.42	0.39	0.36	0.41	0.37	0.33	0.31	0.42	0.4	0.36	0.43	0.38	0.34	0.31
%Ni: tailings							0.21							0.23							0.21
Chalcopyrite: concentrates	3.539127	1.43478	1.391301	0.785942	0.34696	0.244243	0.184509	3.569595	1.426705	1.184214	0.793884	0.329841	0.264902	0.16278	3.473379	1.768095	1.129064	0.878298	0.308994	0.206844	0.157821
Chalcopyrite: tailings							2.484653							2.941942							2.19124
pentlandite: concentrates	0.194703	0.112837	0.276448	0.129966	0.181956	0.188944	0.200633	0.163912	0.15562	0.317606	0.159311	0.1868	0.204423	0.157338	0.146029	0.171582	0.313927	0.15662	0.190092	0.211445	0.216774
pentlandite: tailings							4.392577							4.746954							4.304285
Mass cumulative chalcopyrite	**3.539127**	**4.973908**	**6.365208**	**7.15115**	**7.49811**	**7.742353**	**7.926861**	**3.569595**	**4.996301**	**6.180514**	**6.974399**	**7.30424**	**7.569142**	**7.731922**	**3.473379**	**5.241474**	**6.370538**	**7.248835**	**7.557829**	**7.764673**	**7.922494**
Mass cumulative pentlandite	**0.194703**	**0.30754**	**0.583988**	**0.713954**	**0.89591**	**1.084854**	**1.285487**	**0.163912**	**0.319533**	**0.637139**	**0.79645**	**0.983251**	**1.187674**	**1.345012**	**0.146029**	**0.317611**	**0.631538**	**0.788158**	**0.978251**	**1.189696**	**1.40647**
Cumulative chalcopyrite (%)	33.99243	47.77314	61.13624	68.68502	72.01748	74.36337	76.13553	33.44239	46.80873	57.90325	65.3409	68.43107	70.91285	72.43789	34.34319	51.82531	62.98898	71.67318	74.72838	76.77356	78.33402
Cumulative pentlandite (%)	3.429042	5.416286	10.28498	12.5739	15.77844	19.10606	22.63953	2.690632	5.245152	10.45867	13.07378	16.14012	19.49574	22.07846	2.557091	5.561624	11.05874	13.8013	17.12997	20.83255	24.62844
Cumulative chalc. grade (%)	19.01734	17.06315	10.77935	10.04516	8.251469	6.808858	5.649534	22.25434	15.40167	8.996382	8.237155	6.928704	5.783268	5.095507	24.30636	16.42066	9.40162	8.762039	7.317097	6.026134	5.027282
Cumulative pent. grade (%)	1.046229	1.055026	0.988972	1.002885	0.985925	0.954053	0.916176	1.021898	0.984996	0.927422	0.940652	0.932698	0.907453	0.886393	1.021898	0.995021	0.932021	0.952687	0.947091	0.923318	0.892487

Weight of parameter (g)	30: 3SPW_2							31: 5SPW_1							32: 5SPW_2						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	21.38	26.56	40.98	23.91	31.8	34.65	34.08	22.25	26.05	44.96	23.73	30.21	35.36	35.24	22.97	29.5	48.96	25.74	30.13	36.64	34.75
Paper	7.59	7.73	7.42	7.61	7.7	7.24	7.55	7.66	7.56	7.45	7.68	7.58	7.54	7.66	7.65	7.62	7.55	7.5	7.52	7.61	7.17
D + C + H2O	666.1	697	1051.6	773	1002.4	1116	1379.8	673.3	695.5	1122.4	796	1003.9	1159.4	1453.1	683.5	730.9	1169.9	822.6	1012.8	1190.4	1446.2
Ore recovered	**13.79**	**18.83**	**33.56**	**16.3**	**24.1**	**27.41**	**26.53**	**14.59**	**18.49**	**37.51**	**16.05**	**22.63**	**27.82**	**27.58**	**15.32**	**21.88**	**41.41**	**18.24**	**22.61**	**29.03**	**27.58**
Water recovered	**111.61**	**226.67**	**573.64**	**217.6**	**437.9**	**647.69**	**813.97**	**118.01**	**225.51**	**640.49**	**240.85**	**440.87**	**690.68**	**886.22**	**127.48**	**257.52**	**684.09**	**265.26**	**449.79**	**720.47**	**879.32**
Cumulative ore recovered	13.79	32.62	66.18	82.48	106.58	133.99	160.52	14.59	33.08	70.59	86.64	109.27	137.09	164.67	15.32	37.2	78.61	96.85	119.46	148.49	176.07
Cumulative water recovered	111.61	338.28	911.92	1129.52	1567.42	2215.11	3029.08	118.01	343.52	984.01	1224.86	1665.73	2356.41	3242.63	127.48	385	1069.09	1334.35	1784.14	2504.61	3383.93
%Cu: concentrates	7.99	3.04	1.05	2.04	0.5	0.29	0.21	7.63	3.22	0.98	1.46	0.44	0.26	0.18	7.33	2.69	0.85	1.74	0.46	0.25	0.17
%Cu: tailings							0.09							0.09							0.1
%Ni: concentrates	0.41	0.4	0.35	0.44	0.37	0.34	0.31	0.4	0.39	0.35	0.41	0.35	0.32	0.3	0.4	0.38	0.34	0.42	0.37	0.32	0.29
%Ni: tailings							0.21							0.21							0.24
Chalcopyrite: concentrates	3.184454	1.654428	1.018439	0.96104	0.348266	0.229737	0.16102	3.21739	1.720746	1.062422	0.677254	0.28778	0.209052	0.14348	3.245538	1.701075	1.017298	0.917272	0.300595	0.209754	0.135509
Chalcopyrite: tailings							2.183618							2.172824							2.381301
pentlandite: concentrates	0.137564	0.18326	0.285791	0.174501	0.216959	0.226749	0.200105	0.141995	0.175453	0.319428	0.160109	0.192713	0.216603	0.201314	0.1491	0.202297	0.342564	0.186394	0.203545	0.226024	0.194603
pentlandite: tailings							4.289314							4.268109							4.81127
Mass cumulative chalcopyrite	**3.184454**	**4.838882**	**5.857321**	**6.818361**	**7.166627**	**7.396364**	**7.557384**	**3.21739**	**4.938136**	**6.000558**	**6.677812**	**6.965592**	**7.174645**	**7.318124**	**3.245538**	**4.946613**	**5.96391**	**6.881182**	**7.181777**	**7.391532**	**7.52704**
Mass cumulative pentlandite	**0.137564**	**0.320825**	**0.606616**	**0.781117**	**0.998075**	**1.224825**	**1.424929**	**0.141995**	**0.317448**	**0.636876**	**0.796985**	**0.989698**	**1.206302**	**1.407616**	**0.1491**	**0.351397**	**0.693961**	**0.880355**	**1.0839**	**1.309925**	**1.504528**
Cumulative chalcopyrite (%)	32.69123	49.67539	60.13057	69.9965	73.57176	75.93021	77.58323	33.89957	52.02995	63.22401	70.3598	73.39196	75.5946	77.10636	32.75561	49.92372	60.19081	69.44838	72.48214	74.59909	75.96671
Cumulative pentlandite (%)	2.407396	5.614476	10.61585	13.66965	17.46645	21.43459	24.93645	2.501797	5.593077	11.22105	14.042	17.43739	21.2537	24.80063	2.360743	5.563772	10.9877	13.93894	17.16173	20.74044	23.82166
Cumulative chalc. grade (%)	23.09249	14.83409	8.850591	8.266684	6.724176	5.520087	4.708064	22.05202	14.92786	8.500578	7.707539	6.374661	5.233529	4.444115	21.18497	13.29735	7.586707	7.104989	6.011868	4.977798	4.275027
Cumulative pent. grade (%)	0.997567	0.983522	0.916615	0.947038	0.936457	0.914117	0.887696	0.973236	0.959636	0.902218	0.919882	0.905737	0.879934	0.85481	0.973236	0.944614	0.88279	0.908988	0.907333	0.882163	0.854506

B.2. All Synthetic and Actual Plant Waters at 100 g/t SIPX, 5 g/t Aerophine: Run 1

Weight of parameter (g)	1: Process water_1							3: Process water_2							11: Cu thickener overflow_1						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	23.51	21.52	38.71	15.75	18.99	22.32	20.26	22.44	24.88	42.8	18.01	20.16	22.71	26.63	21.26	19.58	27.5	13.63	11.94	20.01	27.1
Paper	7.57	7.43	7.78	7.62	7.78	7.6	1.94	7.7	7.74	7.52	7.57	7.69	7.69	7.65	7.54	7.46	7.63	7.41	7.63	7.62	7.48
D + C + H_2O	616.5	559.3	751.9	611.2	629.1	610.5	763.2	622.1	577	791.1	631.1	673.4	627.8	801.6	632.7	572.4	709.1	602.4	658	597.1	808.7
Ore recovered	**15.94**	**14.09**	**30.93**	**8.13**	**11.21**	**14.72**	**18.32**	**14.74**	**17.14**	**35.28**	**10.44**	**12.47**	**15.02**	**18.98**	**13.72**	**12.12**	**19.87**	**6.22**	**4.31**	**12.39**	**19.62**
Water recovered	**59.86**	**93.71**	**276.57**	**63.97**	**77.49**	**154.88**	**205.58**	**66.66**	**108.36**	**311.42**	**81.56**	**120.53**	**171.88**	**243.32**	**78.28**	**108.78**	**244.83**	**57.08**	**113.29**	**143.81**	**249.78**
Cumulative ore recovered	15.94	30.03	60.96	69.09	80.3	95.02	113.34	14.74	31.88	67.16	77.6	90.07	105.09	124.07	13.72	25.84	45.71	51.93	56.24	68.63	88.25
Cumulative water recovered	59.86	153.57	430.14	494.11	571.6	726.48	932.06	66.66	175.02	486.44	568	688.53	860.41	1103.73	78.28	187.06	431.89	488.97	602.26	746.07	995.85
Residual concentrate SIPX (mg/L)	N/A	N/A	N/A	24.41	24.2	20.12	18.09	N/A	N/A	N/A	26.29	25.51	23.25	18.57	N/A	N/A	N/A	25.55	24.83	24.59	24.2
Residual tailings SIPX (mg/L)							16.13							16.2							23.02
%Cu: concentrates	14	7.05	3	3.25	1.31	0.8	0.6	9	4.11	1.57	1.67	0.67	0.44	0.32	10.5	4.89	2.01	2.49	1.01	0.76	0.58
%Cu: tailings							0.17							0.11							0.17
%Ni: concentrates	1.69	1.79	1.49	1.33	1.05	0.72	0.56	1.59	1.41	0.9	0.78	0.58	0.43	0.34	0.83	1.01	0.85	1.04	0.7	0.58	0.47
%Ni: tailings							0.27							0.19							0.21
Chalcopyrite: concentrates	6.449711	2.870939	2.681792	0.763656	0.424425	0.340347	0.317688	3.834104	2.035994	1.600855	0.503896	0.241471	0.191006	0.175538	4.163584	1.712913	1.154298	0.447624	0.125812	0.27215	0.32889
Chalcopyrite: tailings							4.356422							2.784749							4.479697
pentlandite: concentrates	0.6554404	0.613652	1.121307	0.263088	0.286387	0.257869	0.249616	0.570234	0.588015	0.772555	0.198131	0.175976	0.157144	0.157012	0.277071	0.297839	0.410937	0.157392	0.073406	0.174847	0.224365
pentlandite: tailings							5.824774							4.049311							4.658577
Mass cumulative chalcopyrite	**6.449711**	**9.32065**	**12.00244**	**12.7661**	**13.19052**	**13.53087**	**13.84856**	**3.834104**	**5.870098**	**7.470954**	**7.97485**	**8.216321**	**8.407327**	**8.582864**	**4.163584**	**5.876497**	**7.030795**	**7.478419**	**7.604231**	**7.876382**	**8.205272**
Mass cumulative pentlandite	**0.6554404**	**1.269092**	**2.390399**	**2.653487**	**2.939873**	**3.197742**	**3.447358**	**0.570234**	**1.158248**	**1.930803**	**2.128934**	**2.30491**	**2.462054**	**2.619066**	**0.277071**	**0.57491**	**0.985847**	**1.143238**	**1.216645**	**1.391491**	**1.615856**
Cumulative chalc. recovery (%)	35.428279	51.19836	65.92945	70.12421	72.45558	74.3251	76.07016	33.72831	51.6388	65.7214	70.15413	72.27833	73.95859	75.50278	32.82297	46.32646	55.42619	58.95497	59.94679	62.09224	64.685
Cumulative pent. recovery (%)	7.0689291	13.68717	25.78047	28.61787	31.70656	34.48767	37.17978	8.55131	17.36927	28.95461	31.92582	34.56478	36.92133	39.27591	4.415866	9.16274	15.71212	18.22059	19.39051	22.17717	25.75303
Cumulative chalc. grade (%)	40.462428	31.0378	19.68905	18.47749	16.42655	14.24002	12.2186	26.01156	18.41311	11.12411	10.27687	9.12215	8.00012	6.917759	30.34682	22.74186	15.38131	14.40096	13.52104	11.47659	9.297758
Cumulative pent. grade (%)	4.1119221	4.226082	3.921258	3.840623	3.661113	3.365336	3.041607	3.868613	3.63315	2.87493	2.743472	2.559021	2.342805	2.110958	2.019465	2.224884	2.156742	2.201499	2.163309	2.027527	1.830999

Weight of parameter (g)	13: Cu thickener overflow_2							7: Ni thickener overflow_1							9: Ni thickener overflow_2						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	19.8	16.63	22.19	17.07	19.86	22.15	25.05	22.12	24.73	45.17	25.52	25.58	37.28	40.71	22.43	18.26	42.92	16.08	18.18	25.15	31.29
Paper	7.39	7.59	7.6	7.53	7.28	7.52	7.43	7.31	7.6	7.41	7.42	7.45	7.42	7.52	7.42	7.12	7.49	7.42	7.51	7.52	7.43
D + C + H_2O	623.7	533.8	647.8	605.1	673.8	813.8	770.7	612.2	551.6	736.4	675.2	717.5	778.9	962.3	636.9	532.8	744.95	606.6	648.7	720.25	874.35
Ore recovered	**12.41**	**9.04**	**14.59**	**9.54**	**12.58**	**14.63**	**17.62**	**14.81**	**17.13**	**37.76**	**18.1**	**18.13**	**29.86**	**33.19**	**15.01**	**11.14**	**35.43**	**8.66**	**10.67**	**17.63**	**23.86**
Water recovered	**70.59**	**73.26**	**188.81**	**56.46**	**120.82**	**358.27**	**213.78**	**56.69**	**82.97**	**254.24**	**118**	**158.97**	**308.14**	**389.81**	**81.19**	**70.16**	**265.12**	**58.84**	**97.63**	**261.72**	**311.19**
Cumulative ore recovered	12.41	21.45	36.04	45.58	58.16	72.79	90.41	14.81	31.94	69.7	87.8	105.93	135.79	168.98	15.01	26.15	61.58	70.24	80.91	98.54	122.4
Cumulative water recovered	70.59	143.85	332.66	389.12	509.94	868.21	1081.99	56.69	139.66	393.9	511.9	670.87	979.01	1368.82	81.19	151.35	416.47	475.31	572.94	834.66	1145.85
Residual concentrate SIPX (mg/L)	N/A	N/A	N/A	25.82	25.58	24.79	23.01	N/A	N/A	N/A	21	20.95	19.14	15.05	N/A	N/A	N/A	21.49	21.69	21.66	15.49
Residual tailings SIPX (mg/L)							22.84							12.71							13.37
%Cu: concentrates	10.06	5.85	2.88	2.88	1.26	0.9	0.84	5.91	4.29	1.84	1.95	0.8	0.45	0.29	3.82	4.71	2.93	4.4	1.83	1.15	0.76
%Cu: tailings							0.19							0.1							0.14
%Ni: concentrates	0.78	0.97	0.88	0.89	0.69	0.59	0.53	0.93	1.23	0.95	1.08	0.71	0.43	0.29	0.88	1.26	1.17	1.48	1.06	0.79	0.61
%Ni: tailings							0.23							0.16							0.19
Chalcopyrite: concentrates	3.608225	1.528439	1.214428	0.794081	0.458116	0.380549	0.427769	2.529685	2.123922	2.008046	1.020087	0.419191	0.388353	0.278182	1.657173	1.516457	3.000286	1.101272	0.564338	0.585968	0.524092
Chalcopyrite: tailings							4.994858							2.401792							3.550983
pentlandite: concentrates	0.235518	0.213353	0.312389	0.206584	0.211197	0.210017	0.227217	0.335117	0.51265	0.872798	0.47562	0.313195	0.312404	0.234187	0.321382	0.341518	1.008591	0.311844	0.275187	0.338873	0.354127
pentlandite: tailings							5.090163							3.235114							4.057032
Mass cumulative chalcopyrite	**3.608225**	**5.136665**	**6.351092**	**7.145173**	**7.603289**	**7.983838**	**8.411607**	**2.529685**	**4.653607**	**6.661653**	**7.68174**	**8.100931**	**8.489283**	**8.767465**	**1.657173**	**3.17363**	**6.173916**	**7.275188**	**7.839526**	**8.425494**	**8.949587**
Mass cumulative pentlandite	**0.235518**	**0.448871**	**0.76126**	**0.967844**	**1.179041**	**1.389058**	**1.616275**	**0.335117**	**0.847766**	**1.720564**	**2.196185**	**2.50938**	**2.821783**	**3.055971**	**0.321382**	**0.6629**	**1.671491**	**1.983336**	**2.258523**	**2.597397**	**2.951523**
Cumulative chalc. recovery (%)	26.91407	38.31483	47.37336	53.29647	56.7136	59.55215	62.74291	22.64864	41.66443	59.64276	68.77575	72.52882	76.0058	78.4964	13.25678	25.38788	49.38908	58.19885	62.71335	67.40088	71.59343
Cumulative pent. recovery (%)	3.511823	6.693136	11.35119	14.43157	17.58074	20.71231	24.10035	5.326852	13.47568	27.34925	34.90948	39.88787	44.85368	48.57621	4.585567	9.458444	23.8493	28.29878	32.22523	37.06037	42.11315
Cumulative chalc. grade (%)	29.07514	23.94715	17.62234	15.67612	13.07306	10.96832	9.303846	17.08092	14.56984	9.557609	8.749134	7.647438	6.251774	5.188463	11.04046	12.13625	10.02585	10.35761	9.689193	8.550329	7.311754
Cumulative pent. grade (%)	1.89781	2.092639	2.112265	2.123397	2.027238	1.908309	1.787717	2.262774	2.654247	2.468529	2.50135	2.368904	2.07805	1.808481	2.141119	2.534991	2.714341	2.823656	2.791402	2.63588	2.411375

Weight of parameter (g)	23: Mine pit waters_1							25: Mine pit waters_2							45: Raw water_1						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	12.84	14.94	28.96	17.05	20.32	22.77	32.81	19.75	18.94	35.43	16.05	19.19	23.22	28.36	30.28	29.67	51.63	20.18	24.31	26.3	32.94
Paper	7.5	7.6	7.84	7.53	7.32	7.42	7.42	7.59	7.78	7.63	7.52	7.57	7.55	7.62	7.34	7.32	7.44	7.53	7.59	7.36	7.5
D + C + H_2O	572.7	505.5	644.6	612	666.1	614.7	860.1	610.1	545.5	765.3	613.6	662.3	630.4	819.9	711.9	730.4	1229.1	686.6	775.3	730.7	979.5
Ore recovered	**5.34**	**7.34**	**21.12**	**9.52**	**13**	**15.35**	**25.39**	**12.16**	**11.16**	**27.8**	**8.53**	**11.62**	**15.67**	**20.74**	**22.94**	**22.35**	**44.19**	**12.65**	**16.72**	**18.94**	**25.44**
Water recovered	**26.66**	**46.66**	**179.08**	**63.38**	**112.7**	**158.45**	**295.41**	**57.24**	**82.84**	**293.1**	**65.97**	**110.28**	**173.83**	**259.86**	**148.26**	**256.55**	**740.51**	**134.85**	**218.18**	**270.86**	**414.76**
Cumulative ore recovered	5.34	12.68	33.8	43.32	56.32	71.67	97.06	12.16	23.32	51.12	59.65	71.27	86.94	107.68	22.94	45.29	89.48	102.13	118.85	137.79	163.23
Cumulative water recovered	26.66	73.32	252.4	315.78	428.48	586.93	882.34	57.24	140.08	433.18	499.15	609.43	783.26	1043.12	148.26	404.81	1145.32	1280.17	1498.35	1769.21	2183.97
Residual concentrate SIPX (mg/L)	N/A	N/A	N/A	21.93	22.14	21.61	20.09	N/A	N/A	N/A	23.75	24.65	23.21	22.46	N/A	N/A	N/A	20.2	20.08	19.29	16.84
Residual tailings SIPX (mg/L)							17.49							19.57							14.69
%Cu: concentrates	10.55	7.25	3.55	4.12	1.43	0.82	0.45	11.96	5.32	1.74	2.36	0.86	0.51	0.34	9.2846	2.5908	0.9343	2.185	0.5428	0.3177	0.2258
%Cu: tailings							0.13							0.1							0.0734
%Ni: concentrates	0.74	0.94	1.12	1.09	0.92	0.74	0.52	1.44	1.51	1.06	0.92	0.66	0.51	0.42	1.1833	1.0456	0.6449	1.0123	0.5249	0.3719	0.3097
%Ni: tailings							0.22							0.18							0.1659
Chalcopyrite: concentrates	1.628237	1.538006	2.166936	1.133595	0.537283	0.363786	0.330217	4.203283	1.715931	1.398035	0.581815	0.288821	0.230974	0.203803	6.155743	1.673537	1.193258	0.798851	0.262301	0.173909	0.166022
Chalcopyrite: tailings							3.392549							2.57896							1.775113
pentlandite: concentrates	0.096146	0.167873	0.575533	0.252477	0.290998	0.276375	0.321236	0.426044	0.410015	0.716983	0.190939	0.186599	0.194445	0.211942	0.66046	0.568593	0.693385	0.311572	0.213536	0.171382	0.191698
pentlandite: tailings							4.833255							3.907971							3.377619
Mass cumulative chalcopyrite	**1.628237**	**3.166243**	**5.333179**	**6.466775**	**7.004058**	**7.367844**	**7.698061**	**4.203283**	**5.919214**	**7.317249**	**7.899064**	**8.187884**	**8.418858**	**8.622662**	**6.155743**	**7.82928**	**9.022538**	**9.821389**	**10.08369**	**10.2576**	**10.42362**
Mass cumulative pentlandite	**0.096146**	**0.264019**	**0.839552**	**1.092029**	**1.383027**	**1.659401**	**1.980637**	**0.426044**	**0.836058**	**1.553041**	**1.743981**	**1.930579**	**2.125024**	**2.336966**	**0.66046**	**1.229053**	**1.922438**	**2.234009**	**2.447545**	**2.618927**	**2.810625**
Cumulative chalc. recovery (%)	14.68122	28.54886	48.08734	58.30856	63.15304	66.43317	69.41062	37.52388	52.84247	65.32312	70.51714	73.09553	75.1575	76.97691	50.46215	64.18109	73.96291	80.51155	82.66178	84.08741	85.44838
Cumulative pent. recovery (%)	1.411029	3.874723	12.32118	16.02651	20.29716	24.35321	29.06763	6.822228	13.38778	24.86881	27.92631	30.91431	34.02796	37.42177	10.67282	19.86109	31.06597	36.10086	39.55154	42.32101	45.41878
Cumulative chalc. grade (%)	30.49133	24.97037	15.77864	14.92792	12.43618	10.28023	7.931239	34.56647	25.38256	14.31387	13.24235	11.48854	9.683527	8.007673	26.8341	17.287	10.0833	9.616557	8.484384	7.444371	6.385849
Cumulative pent. grade (%)	1.800487	2.082172	2.483883	2.520843	2.455658	2.315336	2.040632	3.50365	3.585156	3.038031	2.923689	2.708824	2.444242	2.170288	2.879075	2.713739	2.148455	2.187417	2.059357	1.900666	1.72188

Weight of parameter (g)	47: Raw water_2						
	C1	C2	C3	C4	C5	C6	C7
C + Paper	33.66	27.8	50.46	20.18	26.8	31.27	38.93
Paper	7.24	7.16	7.4	7.3	7.38	7.39	7.46
D + C + H_2O	741.2	714.7	1215.6	688.4	805.4	791.8	1068.3
Ore recovered	**26.42**	**20.64**	**43.06**	**12.88**	**19.42**	**23.88**	**31.47**
Water recovered	**174.08**	**242.56**	**728.14**	**136.42**	**245.58**	**327.02**	**497.53**
Cumulative ore recovered	26.42	47.06	90.12	103	122.42	146.3	177.77
Cumulative water recovered	174.08	416.64	1144.78	1281.2	1526.78	1853.8	2351.33
Residual concentrate SIPX (mg/L)	N/A	N/A	N/A	20.8	20.24	19.13	15.57
Residual tailings SIPX (mg/L)							13.92
%Cu: concentrates	7.9647	2.8296	0.9156	2.2992	0.5332	0.2978	0.1937
%Cu: tailings							0.076
%Ni: concentrates	1.0936	1.115	0.6457	1.1058	0.5251	0.3462	0.2739
%Ni: tailings							0.1662
Chalcopyrite: concentrates	6.081716	1.687946	1.139472	0.855887	0.29927	0.205534	0.176177
Chalcopyrite: tailings							1.806054
pentlandite: concentrates	0.702991	0.559942	0.676493	0.346538	0.248113	0.20115	0.209723
pentlandite: tailings							3.32493
Mass cumulative chalcopyrite	**6.081716**	**7.769662**	**8.909135**	**9.765022**	**10.06429**	**10.26983**	**10.446**
Mass cumulative pentlandite	**0.702991**	**1.262932**	**1.939425**	**2.285962**	**2.534075**	**2.735225**	**2.944949**
Cumulative chalc. recovery (%)	49.63833	63.41517	72.71542	79.70108	82.14369	83.82123	85.25917
Cumulative pent. recovery (%)	11.21219	20.14285	30.93241	36.45944	40.41666	43.62485	46.96979
Cumulative chalc. grade (%)	23.01936	16.51012	9.885857	9.480604	8.221117	7.019703	5.876134
Cumulative pent. grade (%)	2.660827	2.683664	2.152047	2.219381	2.069985	1.8696	1.656606

Weight of parameter (g)	41: 1SPW_1							43: 1SPW_2							37: 3SPW_1						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	26.25	28.32	47.94	21.37	25.83	29.98	33.6	30.87	27.68	45.14	19.39	24.93	29.66	35.9	22.11	28.88	44.54	19.49	24.56	31.87	31.88
Paper	7.43	7.49	7.41	7.46	7.51	7.53	7.57	7.42	7.56	7.52	7.39	7.48	7.41	7.3	7.48	7.56	7.53	7.45	7.51	7.49	7.54
D + C + H_2O	677.1	698.1	1107.7	675.4	750.8	746.1	948.4	714.5	690.9	1075.4	668.8	752.2	759.6	1022.8	637.8	696.5	1050.5	663.9	748.5	785.1	952.8
Ore recovered	**18.82**	**20.83**	**40.53**	**13.91**	**18.32**	**22.45**	**26.03**	**23.45**	**20.12**	**37.62**	**12**	**17.45**	**22.25**	**28.6**	**14.63**	**21.32**	**37.01**	**12.04**	**17.05**	**24.38**	**24.34**
Water recovered	**117.58**	**225.77**	**622.77**	**122.39**	**192.08**	**282.75**	**383.07**	**150.35**	**219.28**	**593.38**	**117.7**	**194.35**	**296.45**	**454.9**	**82.47**	**223.68**	**569.09**	**112.76**	**191.05**	**319.82**	**389.16**
Cumulative ore recovered	18.82	39.65	80.18	94.09	112.41	134.86	160.89	23.45	43.57	81.19	93.19	110.64	132.89	161.49	14.63	35.95	72.96	85	102.05	126.43	150.77
Cumulative water recovered	117.58	343.35	966.12	1088.51	1280.59	1563.34	1946.41	150.35	369.63	963.01	1080.71	1275.06	1571.51	2026.41	82.47	306.15	875.24	988	1179.05	1498.87	1888.03
Residual concentrate SIPX (mg/L)	N/A	N/A	N/A	25.02	25.06	24.35	21.89	N/A	N/A	N/A	24.67	24.61	23.58	20.48	N/A	N/A	N/A	24.6	24.49	23.24	19.58
Residual tailings SIPX (mg/L)							19.79							18.57							16.22
%Cu: concentrates	10.6	2.95	0.97	1.64	0.49	0.3	0.21	8.99	2.89	1.04	1.94	0.54	0.33	0.21	10.72	2.86	0.86	1.66	0.45	0.25	0.2
%Cu: tailings							0.08							0.07							0.07
%Ni: concentrates	1.08	1.03	0.7	0.61	0.44	0.38	0.33	0.98	1	0.71	0.7	0.47	0.37	0.31	1.15	1.13	0.72	0.76	0.5	0.39	0.34
%Ni: tailings							0.2							0.18							0.18
Chalcopyrite: concentrates	5.7656647	1.775968	1.136246	0.659318	0.259445	0.194653	0.157986	6.092934	1.680543	1.130775	0.672832	0.272341	0.212211	0.173584	4.532763	1.762289	0.919902	0.577642	0.221749	0.176156	0.140694
Chalcopyrite: tailings							1.940139							1.696408							1.718095
pentlandite: concentrates	0.4945401	0.522017	0.690292	0.20645	0.196127	0.207567	0.209	0.559148	0.489538	0.649883	0.20438	0.19955	0.200304	0.215718	0.409355	0.58617	0.64835	0.222637	0.207421	0.231343	0.201353
pentlandite: tailings							4.08326							3.672307							3.719255
Mass cumulative chalcopyrite	**5.7656647**	**7.541633**	**8.677879**	**9.337197**	**9.596642**	**9.791295**	**9.94928**	**6.092934**	**7.773477**	**8.904251**	**9.577084**	**9.849425**	**10.06164**	**10.23522**	**4.532763**	**6.295052**	**7.214954**	**7.792595**	**8.014344**	**8.1905**	**8.331194**
Mass cumulative pentlandite	**0.4945401**	**1.016557**	**1.706849**	**1.913299**	**2.109426**	**2.316993**	**2.525993**	**0.559148**	**1.048686**	**1.698569**	**1.902949**	**2.102499**	**2.302803**	**2.518521**	**0.409355**	**0.995526**	**1.643876**	**1.866513**	**2.073934**	**2.305277**	**2.50663**
Cumulative chalc. recovery (%)	48.494083	63.43147	72.98825	78.53366	80.71582	82.35301	83.6818	51.0654	65.15018	74.6273	80.26637	82.54888	84.32744	85.78226	45.10531	62.64177	71.79566	77.54375	79.75036	81.50328	82.90331
Cumulative pent. recovery (%)	7.4825422	15.38082	25.82514	28.9488	31.91625	35.0568	38.21903	9.031885	16.93935	27.43687	30.7382	33.96152	37.19701	40.68149	6.575052	15.9901	26.40389	29.97989	33.31147	37.0273	40.26142
Cumulative chalc. grade (%)	30.635838	19.02051	10.823	9.923686	8.537178	7.26034	6.183902	25.98266	17.84135	10.96718	10.27694	8.902228	7.571402	6.33799	30.98266	17.51058	9.888917	9.167759	7.85335	6.478288	5.525764
Cumulative pent. grade (%)	2.6277372	2.563826	2.128772	2.033478	1.876546	1.718073	1.570012	2.384428	2.4069	2.092092	2.04201	1.900306	1.732864	1.559552	2.798054	2.769195	2.253119	2.195898	2.032273	1.823363	1.662552

Weight of parameter (g)	39: 3SPW_2							33: 5SPW_1							35: 5SPW_2						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	30.26	26.99	45	20.23	26.17	32.03	35.73	24.57	21.92	43.06	17.66	23.07	27.28	31.91	25.96	26.98	45.46	20.81	24.94	33.88	36.98
Paper	7.67	7.53	7.73	7.38	7.61	7.5	7.53	7.38	7.46	7.46	7.51	7.38	7.46	7.54	7.65	7.58	7.63	7.37	7.72	7.47	7.64
D + C + H_2O	689.1	661.5	994.6	665.4	765.4	780.9	1002.1	653	594.9	939.1	641.5	728.2	714.8	930.3	661.8	642.9	970.7	664.6	735.6	781.4	994.6
Ore recovered	**22.59**	**19.46**	**37.27**	**12.85**	**18.56**	**24.53**	**28.2**	**17.19**	**14.46**	**35.6**	**10.15**	**15.69**	**19.82**	**24.37**	**18.31**	**19.4**	**37.83**	**13.44**	**17.22**	**26.41**	**29.34**
Water recovered	**125.81**	**190.54**	**512.93**	**113.45**	**206.44**	**315.47**	**434.6**	**95.11**	**128.94**	**459.1**	**92.25**	**172.11**	**254.08**	**366.63**	**102.79**	**172**	**488.47**	**112.06**	**177.98**	**314.09**	**425.96**
Cumulative ore recovered	22.59	42.05	79.32	92.17	110.73	135.26	163.46	17.19	31.65	67.25	77.4	93.09	112.91	137.28	18.31	37.71	75.54	88.98	106.2	132.61	161.95
Cumulative water recovered	125.81	316.35	829.28	942.73	1149.17	1464.64	1899.24	95.11	224.05	683.15	775.4	947.51	1201.59	1568.22	102.79	274.79	763.26	875.32	1053.3	1367.39	1793.35
Residual concentrate SIPX (mg/L)	N/A	N/A	N/A	24.63	24.35	23.21	19.2	N/A	N/A	N/A	24.39	24.15	23.23	18.34	N/A	N/A	N/A	24.35	23.92	22.79	19.77
Residual tailings SIPX (mg/L)							14.62							17.74							16.46
%Cu: concentrates	8.13	3.3	1.03	1.86	0.52	0.3	0.21	9.65	3.47	1.06	1.72	0.5	0.29	0.23	8.98	3.09	0.94	1.46	0.46	0.26	0.21
%Cu: tailings							0.08							0.08							0.08
%Ni: concentrates	0.99	1.13	0.73	0.72	0.48	0.39	0.33	1.17	1.25	0.86	0.84	0.6	0.44	0.36	1.17	1.19	0.78	0.71	0.49	0.38	0.32
%Ni: tailings							0.19							0.18							0.18
Chalcopyrite: concentrates	5.307997	1.856012	1.109483	0.69078	0.278936	0.212688	0.171156	4.794321	1.450179	1.090636	0.504566	0.226734	0.166121	0.161997	4.752133	1.732543	1.027751	0.567121	0.228936	0.198457	0.178075
Chalcopyrite: tailings							1.934197							1.994728							1.937688
pentlandite: concentrates	0.544139	0.535032	0.661973	0.225109	0.216759	0.232766	0.226423	0.48935	0.439781	0.744915	0.207445	0.229051	0.212185	0.21346	0.521234	0.561703	0.717942	0.232175	0.205299	0.24418	0.228438
pentlandite: tailings							3.867217							3.778336							3.670292
Mass cumulative chalcopyrite	**5.307997**	**7.164009**	**8.273491**	**8.964272**	**9.243208**	**9.455896**	**9.627052**	**4.794321**	**6.2445**	**7.335136**	**7.839702**	**8.066436**	**8.232558**	**8.394555**	**4.752133**	**6.484676**	**7.512428**	**8.079549**	**8.308486**	**8.506942**	**8.685017**
Mass cumulative pentlandite	**0.544139**	**1.07917**	**1.741144**	**1.966253**	**2.183012**	**2.415779**	**2.642202**	**0.48935**	**0.929131**	**1.674046**	**1.881491**	**2.110543**	**2.322727**	**2.536187**	**0.521234**	**1.082937**	**1.800878**	**2.033054**	**2.238353**	**2.482533**	**2.710971**
Cumulative chalc. recovery (%)	45.91197	61.9657	71.56227	77.53723	79.94991	81.78957	83.27	46.14679	60.10521	70.60291	75.45951	77.6419	79.24086	80.80014	44.73562	61.04543	70.72048	76.05924	78.2144	80.08263	81.759
Cumulative pent. recovery (%)	8.359252	16.5786	26.74807	30.20628	33.53621	37.11205	40.59045	7.749601	14.7142	26.51105	29.79626	33.42363	36.78389	40.16435	8.168189	16.97057	28.22135	31.85974	35.07696	38.90347	42.4833
Cumulative chalc. grade (%)	23.49711	17.03688	10.43052	9.725802	8.347519	6.990903	5.889546	27.89017	19.72986	10.90727	10.12881	8.665202	7.291257	6.114915	25.95376	17.19617	9.944967	9.080186	7.823433	6.415008	5.362777
Cumulative pent. grade (%)	2.408759	2.566398	2.195088	2.13329	1.971473	1.786026	1.616421	2.846715	2.935644	2.489288	2.430868	2.267207	2.057149	1.847456	2.846715	2.87175	2.384006	2.284843	2.107677	1.872056	1.673955

B.3. All Synthetic and Actual Plant Waters at Residual SIPX, 100 g/t fresh SIPX, 5 g/t Aerophine: Run 2

Weight of parameter (g)	2: Process water_1							4: Process water_2							12: Cu thickener overflow_1						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	23.9	24.84	25.95	18.45	21.42	25.88	34.01	14.47	23.91	35.05	22.48	24.91	24.69	24.72	27.15	14.73	26.78	18.51	19.23	26.13	28.77
Paper	7.6	7.53	7.6	7.51	7.7	7.51	7.46	7.53	7.36	7.49	7.34	7.35	7.59	7.47	7.31	7.47	7.5	7.37	7.44	7.43	7.43
D + C + H_2O	624.6	556.6	580.6	608.1	655.8	628.8	843.5	574	540.1	638.7	639.4	689.4	627.4	757.9	648.8	502.2	621.3	625.4	661.3	662.5	830.7
Ore recovered	**16.3**	**17.31**	**18.35**	**10.94**	**13.72**	**18.37**	**26.55**	**6.94**	**16.55**	**27.56**	**15.14**	**17.56**	**17.1**	**17.25**	**19.84**	**7.26**	**19.28**	**11.14**	**11.79**	**18.7**	**21.34**
Water recovered	**67.6**	**87.79**	**117.85**	**58.06**	**101.68**	**169.53**	**277.65**	**26.36**	**72.05**	**166.74**	**85.16**	**131.44**	**169.4**	**201.35**	**88.26**	**43.44**	**157.62**	**75.16**	**109.11**	**202.9**	**270.06**
Cumulative ore recovered	16.3	33.61	51.96	62.9	76.62	94.99	121.54	6.94	23.49	51.05	66.19	83.75	100.85	118.1	19.84	27.1	46.38	57.52	69.31	88.01	109.35
Cumulative water recovered	67.6	155.39	273.24	331.3	432.98	602.51	880.16	26.36	98.41	265.15	350.31	481.75	651.15	852.5	88.26	131.7	289.32	364.48	473.59	676.49	946.55
Residual concentrate SIPX (mg/L)	N/A	N/A	N/A	23.22	24.32	21.92	16.65	N/A	N/A	N/A	24.8	24.8	22.8	16.68	N/A	N/A	N/A	26.27	26.42	25.79	25.27
Residual tailings SIPX (mg/L)							14.81							11.98							24.6
%Cu: concentrates	3.46	3.61	2.97	4.42	1.95	0.88	0.47	6.21	3.99	2.84	3.06	1.44	0.8	0.56	3.21	5.25	3.16	4.16	1.83	1.13	0.8
%Cu: tailings							0.11							0.16							0.15
%Ni: concentrates	0.68	0.96	1.1	1.44	1.13	0.69	0.42	0.81	0.95	1.09	1.17	0.98	0.64	0.47	0.78	1.32	1.19	1.63	1.06	0.68	0.44
%Ni: tailings							0.17							0.21							0.14
Chalcopyrite: concentrates	1.63	1.806043	1.57513	1.397538	0.773237	0.467214	0.36065	1.24559	1.908512	2.26215	1.338971	0.730821	0.395376	0.279191	1.840647	1.10159	1.760832	1.339376	0.623575	0.610723	0.49341
Chalcopyrite: tailings							2.792792							4.07815							3.861199
pentlandite: concentrates	0.2696837	0.404321	0.491119	0.383299	0.377217	0.308401	0.271314	0.136774	0.382543	0.73091	0.430993	0.418706	0.266277	0.197263	0.376526	0.233168	0.558229	0.441805	0.304073	0.309392	0.228457
pentlandite: tailings							3.633533							4.506058							3.033844
Mass cumulative chalcopyrite	**1.63**	**3.436043**	**5.011173**	**6.408711**	**7.181948**	**7.649162**	**8.009812**	**1.24559**	**3.154101**	**5.416251**	**6.755223**	**7.486043**	**7.881419**	**8.16061**	**1.840647**	**2.942237**	**4.703069**	**6.042445**	**6.66602**	**7.276743**	**7.770153**
Mass cumulative pentlandite	**0.2696837**	**0.674005**	**1.165124**	**1.548423**	**1.92564**	**2.234041**	**2.505355**	**0.136774**	**0.519316**	**1.250226**	**1.681219**	**2.099925**	**2.366202**	**2.563465**	**0.376526**	**0.609693**	**1.167922**	**1.609727**	**1.9138**	**2.223192**	**2.45165**
Cumulative chalc. recovery (%)	15.088954	31.80755	46.38857	59.32561	66.48349	70.8085	74.14705	10.17742	25.77141	44.2549	55.19532	61.16668	64.3972	66.6784	15.82488	25.29574	40.43441	51.94963	57.31079	62.56145	66.80352
Cumulative pent. recovery (%)	4.3930382	10.97927	18.9794	25.22319	31.36789	36.39163	40.81122	1.934695	7.345846	17.68473	23.78122	29.70391	33.47046	36.26079	6.864023	11.11465	21.2911	29.34517	34.88839	40.52857	44.69333
Cumulative chalc. grade (%)	10	10.22328	9.644291	10.18873	9.373464	8.052597	6.590268	17.94798	13.42742	10.6097	10.20581	8.938559	7.814992	6.909915	9.277457	10.85696	10.1403	10.50495	9.617689	8.268086	7.105764
Cumulative pent. grade (%)	1.6545012	2.00537	2.242348	2.461722	2.513234	2.35187	2.061342	1.970803	2.210797	2.449023	2.539989	2.507373	2.346259	2.170588	1.89781	2.249791	2.518159	2.798553	2.761218	2.526068	2.242021

Weight of parameter (g)	14: Cu thickener overflow_2							8: Ni thickener overflow_1							10: Ni thickener overflow_2						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	15.04	18.09	26.85	17.74	20.03	23.51	27.14	30.65	21.2	33.81	23.33	29.52	27.77	32.63	36.46	20.12	27.81	25.56	29.37	23.52	33.99
Paper	7.64	7.85	7.62	7.69	7.61	7.48	7.48	7.37	7.43	7.5	7.52	7.48	7.41	7.43	7.51	7.69	7.59	7.47	7.51	7.64	7.58
D + C + H_2O	585.4	523.9	620.5	615	665.6	633	807.7	645.1	528.2	633.6	649	742.1	670.6	864.5	594.1	540.3	632.9	755.65	662.5	635.5	769.3
Ore recovered	**7.4**	**10.24**	**19.23**	**10.05**	**12.42**	**16.03**	**19.66**	**23.28**	**13.77**	**26.31**	**15.81**	**22.04**	**20.36**	**25.2**	**28.95**	**12.43**	**20.22**	**18.09**	**21.86**	**15.88**	**26.41**
Water recovered	**37.3**	**62.16**	**156.87**	**65.85**	**112.78**	**176.07**	**248.74**	**81.12**	**62.93**	**162.89**	**94.09**	**179.66**	**209.34**	**300**	**24.45**	**76.37**	**168.28**	**198.46**	**100.24**	**178.72**	**203.59**
Cumulative ore recovered	7.4	17.64	36.87	46.92	59.34	75.37	95.03	23.28	37.05	63.36	79.17	101.21	121.57	146.77	28.95	41.38	61.6	79.69	101.55	117.43	143.84
Cumulative water recovered	37.3	99.46	256.33	322.18	434.96	611.03	859.77	81.12	144.05	306.94	401.03	580.69	790.03	1090.03	24.45	100.82	269.1	467.56	567.8	746.52	950.11
Residual concentrate SIPX (mg/L)	N/A	N/A	N/A	26.39	26.35	26.31	25.1	N/A	N/A	N/A	20.78	20.57	19.34	15.42	N/A	N/A	N/A	21.92	22.01	21.66	16.53
Residual tailings SIPX (mg/L)							24.19							13.24							14.58
%Cu: concentrates	6.16	5.14	3.26	4.63	1.87	1.14	0.75	2.65	3.67	2.38	2.74	1.08	0.6	0.39	5.22	4.89	2.83	4.26	1.74	0.87	0.6
%Cu: tailings							0.13							0.1							0.12
%Ni: concentrates	1.06	1.26	1.23	1.78	1.09	0.67	0.45	0.61	0.99	0.98	1.26	0.78	0.5	0.36	1.1	1.4	1.24	1.88	1.13	0.65	0.48
%Ni: tailings							0.14							0.14							0.15
Chalcopyrite: concentrates	1.317457	1.521202	1.811844	1.344841	0.671254	0.528156	0.426156	1.783006	1.460575	1.809763	1.252006	0.687954	0.353064	0.284046	4.367601	1.756725	1.653832	2.227266	1.099318	0.399295	0.457977
Chalcopyrite: tailings							3.400176							2.465983							2.969341
pentlandite: concentrates	0.190852	0.313927	0.575496	0.435255	0.329387	0.261316	0.215255	0.345518	0.331686	0.627343	0.484686	0.418277	0.247689	0.22073	0.774818	0.423406	0.610044	0.827474	0.601017	0.251144	0.308438
pentlandite: tailings							3.082623							2.90638							3.124672
Mass cumulative chalcopyrite	**1.317457**	**2.838659**	**4.650503**	**5.995344**	**6.666598**	**7.194754**	**7.62091**	**1.783006**	**3.243581**	**5.053344**	**6.30535**	**6.993303**	**7.346367**	**7.630413**	**4.367601**	**6.124327**	**7.778159**	**10.00542**	**11.10474**	**11.50404**	**11.96201**
Mass cumulative pentlandite	**0.190852**	**0.504779**	**1.080275**	**1.51553**	**1.844917**	**2.106234**	**2.321489**	**0.345518**	**0.677204**	**1.304547**	**1.789234**	**2.207511**	**2.4552**	**2.675929**	**0.774818**	**1.198224**	**1.808268**	**2.635742**	**3.236759**	**3.487903**	**3.796341**
Cumulative chalc. recovery (%)	11.95396	25.75662	42.19641	54.39885	60.48948	65.28171	69.14845	17.65982	32.12613	50.05097	62.45149	69.26534	72.76227	75.57561	29.2512	41.01655	52.09279	67.00949	74.37197	77.04617	80.11339
Cumulative pent. recovery (%)	3.531599	9.340639	19.98987	28.04402	34.13914	38.97465	42.95783	6.189522	12.13126	23.36932	32.05185	39.54476	43.98179	47.93589	11.19515	17.31284	26.12721	38.08319	46.76714	50.39585	54.85239
Cumulative chalc. grade (%)	17.80347	16.09217	12.61324	12.7778	11.23458	9.545913	8.019478	7.65896	8.754604	7.975606	7.964317	6.909696	6.042911	5.198892	15.08671	14.80021	12.62688	12.55543	10.93525	9.796506	8.316195
Cumulative pent. grade (%)	2.579075	2.861557	2.929956	3.230031	3.109062	2.794525	2.442901	1.484185	1.827812	2.058945	2.259989	2.181119	2.019577	1.823213	2.676399	2.895659	2.935499	3.307494	3.187355	2.970197	2.63928

Weight of parameter (g)	24: Mine pit waters_1							26: Mine pit waters_2							46: Raw water_1						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	16.25	16.64	30.94	19.44	24.19	30.1	32.2	23.66	16.32	29.74	16.42	24.69	28.33	33.4	17.84	27.85	44.6	27.24	19.69	34.31	37.79
Paper	7.48	7.58	7.6	7.66	7.65	7.47	7.51	7.54	7.5	7.63	7.67	7.65	7.68	7.58	7.49	7.38	7.44	7.41	7.24	7.31	7.39
D + C + H_2O	593.8	516	658.2	628.5	710	715.8	892.9	635.2	516.6	650.3	609.8	710.3	700.8	920.7	597.8	598.5	773.8	637.9	746.6	779.2	1002.7
Ore recovered	**8.77**	**9.06**	**23.34**	**11.78**	**16.54**	**22.63**	**24.69**	**16.12**	**8.82**	**22.11**	**8.75**	**17.04**	**20.65**	**25.82**	**10.35**	**20.47**	**37.16**	**19.83**	**12.45**	**27**	**30.4**
Water recovered	**44.33**	**55.44**	**190.46**	**77.62**	**153.06**	**252.27**	**328.91**	**78.38**	**56.28**	**183.79**	**61.95**	**152.86**	**239.25**	**355.58**	**46.75**	**126.53**	**292.24**	**78.97**	**193.75**	**311.3**	**433**
Cumulative ore recovered	8.77	17.83	41.17	52.95	69.49	92.12	116.81	16.12	24.94	47.05	55.8	72.84	93.49	119.31	10.35	30.82	67.98	87.81	100.26	127.26	157.66
Cumulative water recovered	44.33	99.77	290.23	367.85	520.91	773.18	1102.09	78.38	134.66	318.45	380.4	533.26	772.51	1128.09	46.75	173.28	465.52	544.49	738.24	1049.54	1482.54
Residual concentrate SIPX (mg/L)	N/A	N/A	N/A	16.98	16.83	16.7	14.47	N/A	N/A	N/A	16.83	16.98	16.7	14.47	N/A	N/A	N/A	20.19	20.23	19.98	18.31
Residual tailings SIPX (mg/L)							11.95							11.95							16.62
%Cu: concentrates	6.59	5.35	2.83	3.7	1.34	0.66	0.43	3.92	5.16	2.77	4.68	1.61	0.79	0.46	8.7686	4.0205	1.7582	3.2445	1.0688	0.5318	0.3439
%Cu: tailings							0.11							0.12							0.092
%Ni: concentrates	1.15	1.44	1.23	1.52	0.92	0.53	0.36	0.86	1.32	1.15	1.65	1.05	0.59	0.38	1.007	0.9672	0.8307	1.0991	0.7732	0.4911	0.3508
%Ni: tailings							0.14							0.15							0.1269
Chalcopyrite: concentrates	1.670355	1.400896	1.909023	1.259711	0.640566	0.431671	0.306841	1.826312	1.315353	1.770078	1.183526	0.792902	0.471488	0.343272	2.622977	2.378602	1.888286	1.859492	0.384583	0.414988	0.302155
Chalcopyrite: tailings							2.807829							3.054416							2.239748
pentlandite: concentrates	0.245389	0.317431	0.698496	0.435659	0.370238	0.291822	0.216263	0.337304	0.28327	0.61865	0.351277	0.435328	0.296436	0.238725	0.253588	0.481717	0.751066	0.530296	0.234218	0.32262	0.259473
pentlandite: tailings							3.008433							3.214197							2.600802
Mass cumulative chalcopyrite	**1.670355**	**3.071251**	**4.980275**	**6.239986**	**6.880552**	**7.312223**	**7.619064**	**1.826312**	**3.141665**	**4.911743**	**6.095269**	**6.888171**	**7.359659**	**7.702931**	**2.622977**	**5.001579**	**6.889866**	**8.749358**	**9.133941**	**9.548929**	**9.851084**
Mass cumulative pentlandite	**0.245389**	**0.56282**	**1.261316**	**1.696976**	**2.067214**	**2.359036**	**2.575299**	**0.337304**	**0.620574**	**1.239224**	**1.590501**	**2.02583**	**2.322265**	**2.56099**	**0.253588**	**0.735305**	**1.486371**	**2.016667**	**2.250884**	**2.573505**	**2.832977**
Cumulative chalc. recovery (%)	16.01969	29.4551	47.76374	59.84511	65.98852	70.12849	73.07127	16.97735	29.20483	45.65943	56.66145	64.03224	68.41519	71.60623	21.69393	41.36671	56.98422	72.36357	75.54435	78.97661	81.47565
Cumulative pent. recovery (%)	4.394718	10.07964	22.58913	30.39142	37.02208	42.24838	46.12147	5.840575	10.74553	21.45773	27.54025	35.07816	40.21108	44.34471	4.666874	13.53211	27.35428	37.11352	41.42392	47.36123	52.13641
Cumulative chalc. grade (%)	19.04624	17.22519	12.09685	11.78468	9.9015	7.937714	6.522612	11.32948	12.59689	10.43941	10.92342	9.456577	7.872135	6.456232	25.34277	16.22836	10.13514	9.963966	9.110254	7.50348	6.248309
Cumulative pent. grade (%)	2.798054	3.15659	3.063678	3.204864	2.974837	2.56083	2.204691	2.092457	2.488269	2.633845	2.850361	2.781205	2.483972	2.146501	2.450122	2.385805	2.186483	2.296625	2.245047	2.022242	1.79689

Weight of parameter (g)	48: Raw water_2						
	C1	C2	C3	C4	C5	C6	C7
C + Paper	23.44	22.59	46.47	29.26	31.53	38.43	39.81
Paper	7.44	7.35	7.45	7.46	7.26	7.19	7.29
D + C + H_2O	629.8	555.7	779.7	704.4	788	837.4	1019.5
Ore recovered	**16**	**15.24**	**39.02**	**21.8**	**24.27**	**31.24**	**32.52**
Water recovered	**73.1**	**88.96**	**296.28**	**143.5**	**223.33**	**365.26**	**447.68**
Cumulative ore recovered	16	31.24	70.26	92.06	116.33	147.57	180.09
Cumulative water recovered	73.1	162.06	458.34	601.84	825.17	1190.43	1638.11
Residual concentrate SIPX (mg/L)	N/A	N/A	N/A	20.34	20.02	19.5	16.8
Residual tailings SIPX (mg/L)							14.85
%Cu: concentrates	6.5673	4.4137	1.8356	2.1594	0.8196	0.4425	0.3126
%Cu: tailings							0.0971
%Ni: concentrates	0.8943	1.0371	0.8204	0.928	0.6314	0.4199	0.3219
%Ni: tailings							0.1293
Chalcopyrite: concentrates	3.036902	1.944069	2.07009	1.360547	0.574904	0.399529	0.293808
Chalcopyrite: tailings							2.300961
pentlandite: concentrates	0.348146	0.38456	0.778881	0.492224	0.372849	0.319165	0.2547
pentlandite: tailings							2.579425
Mass cumulative chalcopyrite	**3.036902**	**4.980971**	**7.051061**	**8.411608**	**8.986512**	**9.386041**	**9.679849**
Mass cumulative pentlandite	**0.348146**	**0.732706**	**1.511587**	**2.003811**	**2.376659**	**2.695824**	**2.950524**
Cumulative chalc. recovery (%)	25.34805	41.57457	58.85295	70.20901	75.00755	78.34229	80.79461
Cumulative pent. recovery (%)	6.295645	13.24977	27.33455	36.2356	42.97795	48.74952	53.35536
Cumulative chalc. grade (%)	18.98064	15.94421	10.03567	9.137093	7.725017	6.360399	5.375006
Cumulative pent. grade (%)	2.175912	2.345409	2.151419	2.176635	2.043032	1.82681	1.638361

Weight of parameter (g)	42: 1SPW_1							44: 1SPW_2							38: 3SPW_1						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	26.64	23.3	45.42	21.09	30.46	34.98	38.45	20.65	23.15	41.53	23.79	26.62	36.29	34.63	18.63	20.51	44.88	23.74	31.23	33.66	33.91
Paper	7.34	7.45	7.34	7.56	7.36	7.6	7.37	7.51	7.42	7.38	7.5	7.44	7.49	7.45	7.27	7.34	7.51	7.48	7.53	7.58	7.47
D + C + H_2O	654.1	568.2	782.9	646.5	786	807	1044.1	622.3	570.1	763.2	675	750.1	823.4	982.4	597.5	544	792.7	683.6	815.4	804.5	975.3
Ore recovered	19.3	15.85	38.08	13.53	23.1	27.38	31.08	13.14	15.73	34.15	16.29	19.18	28.8	27.18	11.36	13.17	37.37	16.26	23.7	26.08	26.44
Water recovered	94.1	100.85	300.42	93.87	222.5	338.72	473.72	68.46	102.87	284.65	119.61	190.52	353.7	415.92	45.44	79.33	310.93	128.24	251.3	337.52	409.56
Cumulative ore recovered	19.3	35.15	73.23	86.76	109.86	137.24	168.32	13.14	28.87	63.02	79.31	98.49	127.29	154.47	11.36	24.53	61.9	78.16	101.86	127.94	154.38
Cumulative water recovered	94.1	194.95	495.37	589.24	811.74	1150.46	1624.18	68.46	171.33	455.98	575.59	766.11	1119.81	1535.73	45.44	124.77	435.7	563.94	815.24	1152.76	1562.32
Residual concentrate SIPX (mg/L)	N/A	N/A	N/A	24.36	24.99	24	21.1	N/A	N/A	N/A	24.7	25.15	24.08	21.6	N/A	N/A	N/A	23.75	24.16	22.92	18.44
Residual tailings SIPX (mg/L)							19.06							17.48							13.99
%Cu: concentrates	5.56	3.91	1.82	2.85	1.03	0.53	0.33	8.18	4.67	1.89	2.56	1.04	0.5	0.32	5	4.39	1.78	2.3	0.72	0.42	0.3
%Cu: tailings							0.09							0.11							0.1
%Ni: concentrates	0.78	0.89	0.78	1.03	0.73	0.47	0.34	1.05	1.1	0.86	1.04	0.76	0.46	0.32	0.82	1.13	0.95	1.15	0.7	0.45	0.32
%Ni: tailings							0.12							0.15							0.15
Chalcopyrite: concentrates	3.1013873	1.791142	2.003052	1.114465	0.687659	0.419405	0.296428	3.106509	2.123095	1.865419	1.205272	0.576509	0.416185	0.251376	1.641618	1.670991	1.922503	1.080867	0.493179	0.316578	0.229249
Chalcopyrite: tailings							2.163329							2.688101							2.443988
pentlandite: concentrates	0.3662774	0.343224	0.722686	0.339073	0.410292	0.313105	0.257109	0.335693	0.420998	0.714574	0.412204	0.354667	0.322336	0.21162	0.226647	0.362095	0.863783	0.454964	0.40365	0.285547	0.205859
pentlandite: tailings							2.428263							3.085876							3.086204
Mass cumulative chalcopyrite	3.1013873	4.892529	6.895581	8.010046	8.697705	9.11711	9.413538	3.106509	5.229604	7.095023	8.300295	8.876803	9.292988	9.544364	1.641618	3.31261	5.235113	6.31598	6.809159	7.125737	7.354986
Mass cumulative pentlandite	0.3662774	0.709501	1.432187	1.77126	2.181552	2.494657	2.751766	0.335693	0.756691	1.471265	1.88347	2.238136	2.560472	2.772092	0.226647	0.588742	1.452526	1.907489	2.311139	2.596686	2.802545
Cumulative chalc. recovery (%)	26.789521	42.26125	59.56345	69.1901	75.13004	78.75282	81.31334	25.39561	42.75184	58.00158	67.85464	72.56758	75.96987	78.02486	16.75296	33.80568	53.42511	64.45552	69.48849	72.71922	75.05873
Cumulative pent. recovery (%)	7.0709519	13.69686	27.64825	34.19402	42.11467	48.15913	53.1226	5.730544	12.91729	25.11562	32.15227	38.2067	43.70922	47.32174	3.848817	9.997744	24.66611	32.39209	39.24668	44.09571	47.59151
Cumulative chalc. grade (%)	16.069364	13.919	9.416333	9.232418	7.917081	6.643187	5.592644	23.64162	18.11432	11.25837	10.46563	9.012898	7.300643	6.178782	14.45087	13.50432	8.457371	8.080834	6.684821	5.569593	4.764209
Cumulative pent. grade (%)	1.8978102	2.018496	1.955739	2.041563	1.985757	1.817733	1.634842	2.554745	2.621029	2.3346	2.37482	2.27245	2.011526	1.794583	1.995134	2.40009	2.346568	2.440493	2.268936	2.029612	1.815355

Weight of parameter (g)	40: 3SPW_2							34: 5SPW_1							36: 5SPW_2						
	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7	C1	C2	C3	C4	C5	C6	C7
C + Paper	21.85	22.91	40.12	27.54	26.71	32.9	37.43	23.25	22.32	42.53	23.38	31.24	35.99	38.33	14.81	24.2	48.49	24.91	29.05	36.29	38.4
Paper	7.48	7.47	7.56	7.51	7.69	7.35	7.63	7.73	7.73	7.52	7.54	7.4	7.63	7.22	7.56	7.44	7.61	7.55	7.74	7.83	7.35
D + C + H_2O	631.9	558.7	748.6	707.5	747.9	783.1	1025.3	628.6	557.7	762.1	671.1	795.6	803.9	1016.12	584.5	570.4	820.1	678.5	776.7	828.3	1058.3
Ore recovered	14.37	15.44	32.56	20.03	19.02	25.55	29.8	15.52	14.59	35.01	15.84	23.84	28.36	31.11	7.25	16.76	40.88	17.36	21.31	28.46	31.05
Water recovered	76.83	91.76	271.64	148.37	188.48	316.65	456.2	72.38	91.61	282.69	116.16	231.36	334.64	445.71	36.55	102.14	334.82	122.04	214.99	358.94	487.95
Cumulative ore recovered	14.37	29.81	62.37	82.4	101.42	126.97	156.77	15.52	30.11	65.12	80.96	104.8	133.16	164.27	7.25	24.01	64.89	82.25	103.56	132.02	163.07
Cumulative water recovered	76.83	168.59	440.23	588.6	777.08	1093.73	1549.93	72.38	163.99	446.68	562.84	794.2	1128.84	1574.55	36.55	138.69	473.51	595.55	810.54	1169.48	1657.43
Residual concentrate SIPX (mg/L)	N/A	N/A	N/A	23.51	23.81	22.7	18.81	N/A	N/A	N/A	24.62	24.33	22.86	17.63	N/A	N/A	N/A	23.45	23.12	21.73	17.25
Residual tailings SIPX (mg/L)							16.32							15.01							13.53
%Cu: concentrates	6.53	4.32	2.02	2.26	1	0.52	0.34	4.43	3.8	1.79	2.6	0.83	0.45	0.32	7.64	4.2	1.73	2.61	0.92	0.48	0.3
%Cu: tailings							0.11							0.1							0.09
%Ni: concentrates	0.84	1.01	0.92	1	0.79	0.49	0.34	0.79	1.04	0.89	1.2	0.7	0.44	0.33	0.94	1	0.86	1.11	0.73	0.45	0.31
%Ni: tailings							0.15							0.14							0.13
Chalcopyrite: concentrates	2.712026	1.927769	1.900902	1.308318	0.549711	0.383988	0.292832	1.987098	1.60237	1.811211	1.190289	0.571884	0.368844	0.287723	1.600867	2.034451	2.044	1.309526	0.566624	0.394821	0.26922
Chalcopyrite: tailings							2.680789							2.415405							2.176986
pentlandite: concentrates	0.293693	0.379426	0.728837	0.487348	0.365591	0.304611	0.246521	0.298316	0.369187	0.758124	0.462482	0.406034	0.303611	0.249788	0.165815	0.407786	0.855397	0.468847	0.378499	0.311606	0.234197
pentlandite: tailings							3.077482							2.846769							2.647224
Mass cumulative chalcopyrite	2.712026	4.639795	6.540697	7.849014	8.398725	8.782714	9.075546	1.987098	3.589468	5.400679	6.590968	7.162853	7.531697	7.819419	1.600867	3.635318	5.679318	6.988844	7.555468	7.950289	8.219509
Mass cumulative pentlandite	0.293693	0.673119	1.401956	1.889304	2.254895	2.559506	2.806027	0.298316	0.667504	1.425628	1.888109	2.294144	2.597754	2.847543	0.165815	0.573601	1.428998	1.897844	2.276343	2.587949	2.822146
Cumulative chalc. recovery (%)	23.06863	39.46634	55.63551	66.76413	71.44	74.70622	77.19707	19.41507	35.07113	52.76768	64.39748	69.98511	73.58892	76.40013	15.39814	34.96677	54.62724	67.22308	72.67323	76.47086	79.06039
Cumulative pent. recovery (%)	4.991808	11.44078	23.82857	32.11186	38.32569	43.50306	47.69309	5.238848	11.72229	25.036	33.15782	40.28834	45.62016	50.00679	3.031704	10.48751	26.12728	34.69951	41.61984	47.31713	51.59911
Cumulative chalc. grade (%)	18.87283	15.56456	10.48693	9.525503	8.281133	6.917157	5.789084	12.80347	11.92118	8.293426	8.141018	6.834783	5.656125	4.760102	22.08092	15.14085	8.752224	8.497075	7.29574	6.022034	5.040479
Cumulative pent. grade (%)	2.043796	2.258032	2.247805	2.292845	2.223324	2.015835	1.7899	1.922141	2.216884	2.189232	2.332151	2.189068	1.950852	1.733453	2.287105	2.389009	2.202185	2.307409	2.198091	1.96027	1.730635